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MINING METHODS AND COSTS AT FRESNILLO, ZACATECAS, MEXICO

BY

A. LIVINGSTON

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MINING METHODS AND COSTS AT FRESNILLO, ZACATECAS, MEXICO[1]

By A. Livingston[2]

INTRODUCTION

This paper on mining practice at the Fresnillo silver mines is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of North America.

Two classes of ore are produced at Fresnillo. A low-grade oxidized silver ore; and a complex-base sulphide ore containing gold, silver, lead, zinc, and copper. This paper is confined to a discussion of operations relating to the production of oxidized silver ores. To date, more than 10 million tons of this ore have been mined. The present (June, 1931) daily production is 2,000 tons of ore, and approximately 600 men are employed in the mining operations.

ACKNOWLEDGMENTS

Thanks are due to Thomas C. Baker, general manager of The Fresnillo Co., for free use of his article on Fresnillo glory-hole practice,[3] and for many helpful suggestions in the preparation of this paper.

LOCATION

Fresnillo is 35 miles north of the city of Zacatecas, Mexico, and 750 miles south of El Paso, Tex. It is connected with the Mexican Central Railroad by a 5-1/2-mile standard-gage spur owned by the mining company. The town, of 12,000 population, is located in a broad semiarid valley on the Mexican plateau, at an altitude of 7,300 feet above sea level. It is partly surrounded by a number of low hills, the largest of which, called

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6661."

2 - One of the consulting engineers, U. S. Bureau of Mines, and assistant mine superintendent, The Fresnillo Co.

3 - Baker, Thomas C., Fresnillo Glory-hole Mining Practice: Eng. and Min. Jour.-Press, vol. 116, Dec. 1, 1923, pp. 931-942.

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Proaño Hill, lies to the south. In and under Proaño Hill lies an extensive deposit of silver ore which has been the basis of important mining operations for the last 350 years and which, under the present more intensive exploitation, still has a long productive life before it.

HISTORY

The Spanish "conquistadores" reached Zacatecas in the year 1546 and discovered the enormous bodies of silver ore that have placed the Zacatecas district fourth in the history of the world in total production of silver bullion. Though only 35 miles from Zacatecas, the Fresnillo mines were not discovered until 1570. Mining operations begun at that time were carried on intermittently and through many vicissitudes for 250 years. By 1835 most of the richer and easily accessible ore had been exhausted, and the mines had reached a depth where drainage of the workings had become a vital problem with which the many small companies and independent operators were individually unable to cope. The various properties were consolidated by government decree and control of the whole operation passed to an English company.

For the ensuing 30 years the mines experienced a tremendous revival under the influence of an able and progressive management. Two Cornish pumps were brought from England and hauled overland on wagons from Vera Cruz to Fresnillo. They were erected in a very substantial way, with surface buildings, boiler houses, and stacks of cut stone, the latter about 80 feet high.

Two shafts were sunk, one 11 by 19 feet, the other 10 by 14 feet, and reached a total depth of 1,400 feet below the plain. Mining operations on a large scale for the period were successfully carried on at that depth.

During the period, the Patio process for extracting the silver from the ore by amalgamation was brought to a high point of development. Steam-driven arrastres, colloquially called "tahonas," were used to grind the ore. The Hacienda Proaño was constructed, 1,150 feet square and enclosed in walls 18 feet high and 3 feet thick; also a theatre, a church, and a large building to house a School of Mines. All of this construction was done on an impressive scale and in a very substantial manner. The old buildings still survive and are in use, after a hundred years.

Operations on this scale finally ceased in 1867. The immediate cause for the shutdown was inability to secure necessary supplies, chiefly fuel for the boilers, during the period of political and economic disruption following the French occupation of the country. The water level in the mines rose rapidly after the shutdown. From 1867 to 1903 mining operations were of small importance, and consisted of robbing pillars of richer ore above the water level and re-sorting old dumps on the surface.

The Fresnillo Mining Co. entered the district in 1903 and erected a lixiviation plant for the treatment of old tailings. A few years later it extended activities to the mine and in 1911-12 erected a 500-ton cyanide plant to treat the low-grade ore available in dumps and surface workings on

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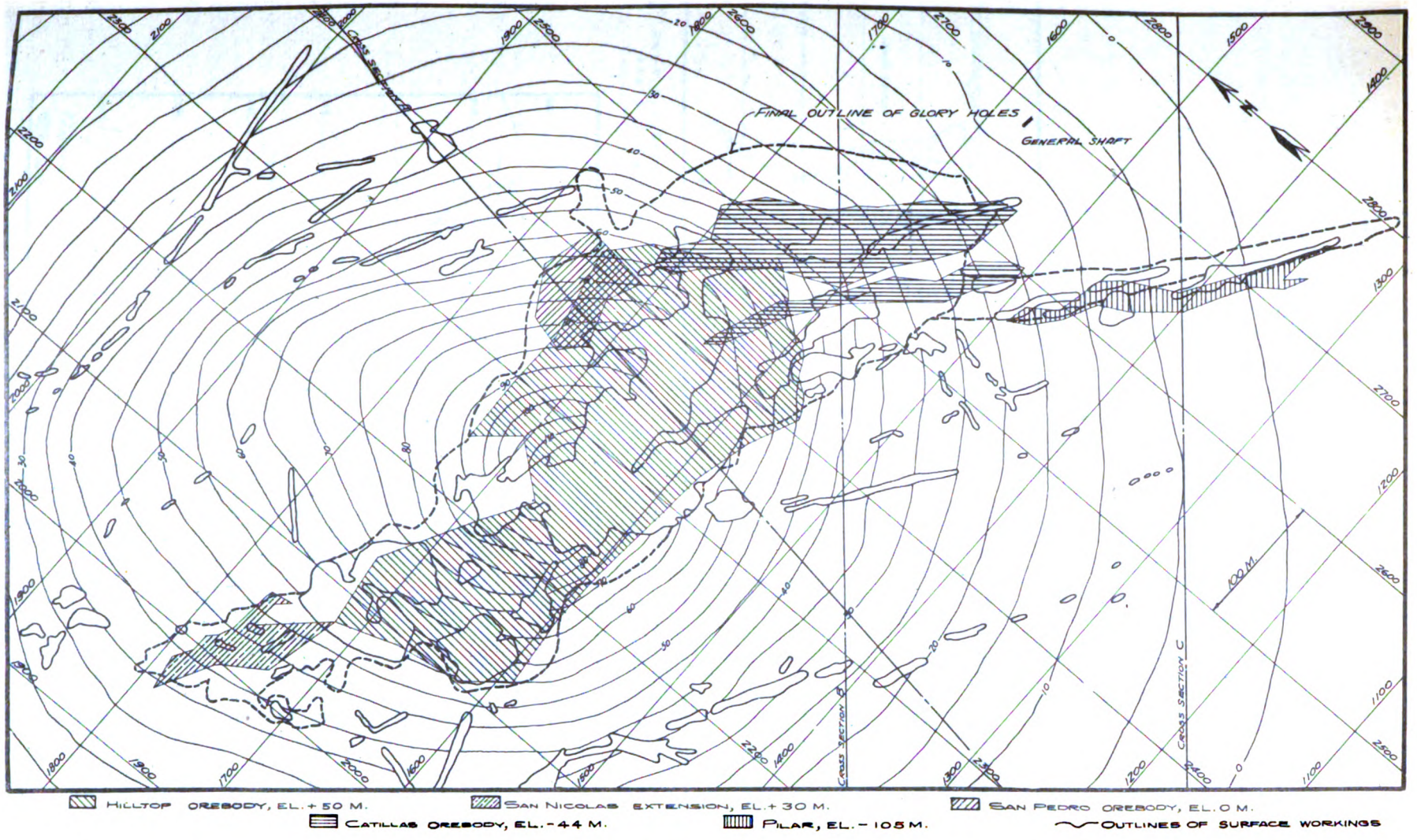
FINAL OUTLINE OF GLORY HOLES
GENERAL SHAFT
CROSS SECTION A
CROSS SECTION B
CROSS SECTION C
100 M.
HILLTOP OREBODY, EL. + 50 M.
SAN NICOLAS EXTENSION, EL. + 30 M.
SAN PEDRO OREBODY, EL. 0 M.
CATILLAS OREBODY, EL. - 44 M.
PILAR, EL. - 105 M.
OUTLINES OF SURFACE WORKINGS

Figure 1.- Plan of orebodies

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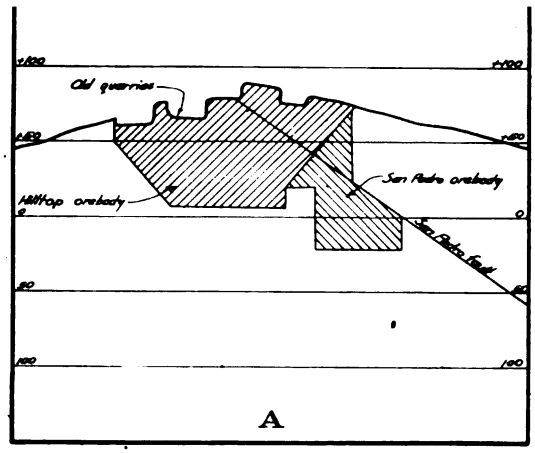

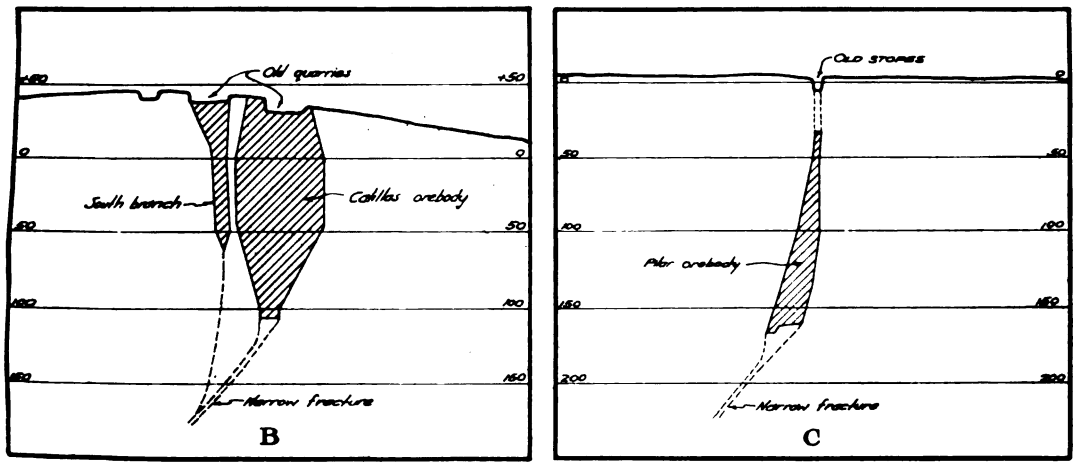

Figure 2.- Cross sections of orebodies: A, Hilltop and San Pedro, looking west; B, Catillas, looking northwest; C, Pilar, looking northwest

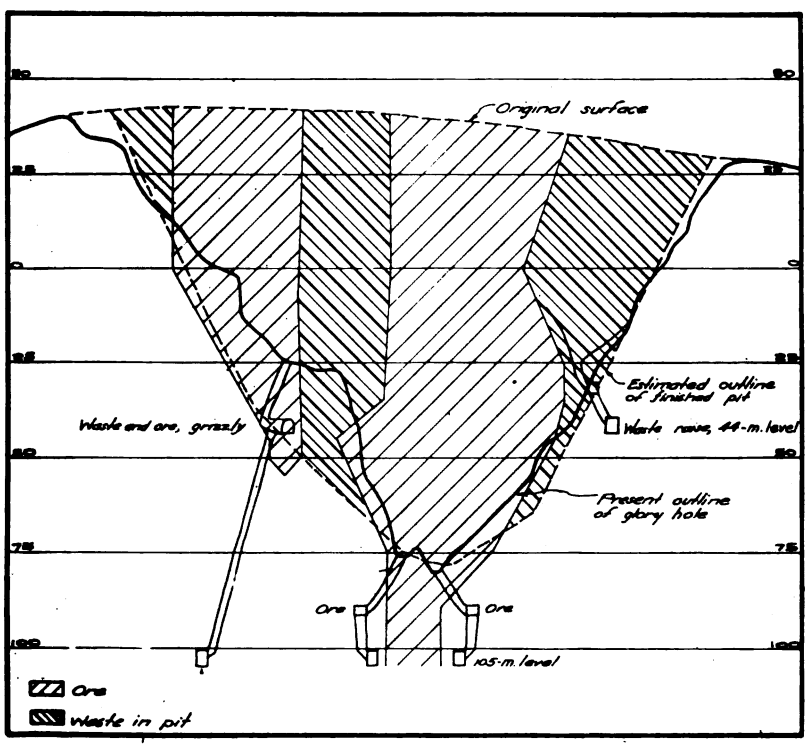

Figure 3.- Typical cross section, Catillas orebody

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Proaño Hill. Operations had to be suspended in 1913 because of a revolutionary upheaval and were not resumed until 1919.

In November of that year the property was leased to the Mexican Corporation, an English company, and a modern, large-scale operation was undertaken. The basis of this operation was the Hilltop orebody, proved by the exploration work of the Fresnillo Mining Co. to contain at least 5,000,000 tons of payable ore. A modern cyanide plant having a capacity of 2,200 tons per day was built to treat this ore. The plant was subsequently enlarged and has treated a maximum of approximately 125,000 tons of ore per month. Thorough exploration of the property added greatly to the ore reserves.

Only very meager data exist as to the production of former operators. However, on the basis of the old tailings left by them, it appears reasonable to conclude that the mine had produced, prior to the advent of the present company, a total of at least 4,000,000 tons of ore. From this at least 20 ounces of silver per ton was recovered, making a total production of at least 80,000,000 ounces of silver. These figures are approximately and probably low.

Since the inception of its operations the present company has produced 10,046,000 tons of ore from which 41,275,000 ounces of silver has been recovered.

Total production from the mines is thus shown to be in excess of 120,000,000 ounces of silver. Gold occurs in the ratio of about 1.4 grams per kilo of silver, or 0.14 per cent.

GEOLOGY

Proaño Hill is dome shaped, rising with gentle slopes to about 325 feet above the plain. Its major axis lies east-west and is about 4,500 feet long. Transversely, its minor axis is about 3,000 feet long. The plan (fig. 1) shows the surface of the hill, the principal surface workings, and the relative locations of the principal orebodies. Sections of the orebodies are shown in Figure 2.

The country rocks are shales and graywackes, probably variants of the same sedimentary series, of Devonian age. No igneous rocks have been positively identified within the developed area, although certain minor variants of the graywacke have been petrologically classified as rhyolite tuff.

The orebodies in Proaño Hill constitute a true stockwork formed in an extensive zone of intense fracturing. On several of the larger fractures the movement, while small, was sufficient to make formal (regular) veins. Faulting of importance was entirely absent. Because of the friable nature of the formations, cross fracturing and shattering took place in every conceivable direction, and the entire fractured zone was subjected to the mineralizing influences. Except in a few of the larger fissure veins the deposition of minerals took place in minute veinlets on the fracture planes

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with little penetration or alteration of the intervening masses. In many superficial aspects the whole deposit bears a resemblance to the porphyry-copper deposits, though, of course, it is fundamentally different from them.

There is very little segregated quartz in the orebodies. Nevertheless, Proaño Hill owes its existence to a pronounced degree of silicification of the country rocks, which very probably was from the same source as the metallization. The principal metallic mineral introduced was pyrite, accompanied by traces of lead, zinc, and copper sulphides, and carrying also the original silver minerals. Manganese in some form was a very important constituent of the mineralization. There is little evidence of secondary sulphide enrichment. The entire stockwork has been thoroughly oxidized, and the ore has a characteristic reddish to blackish color.

The higher oxides of manganese tend to reduce the solubility of the silver in cyanide, and have presented a difficult metallurgical problem. Mill recoveries have averaged between 70 and 75 per cent.

Again referring to Figures 1 and 2, it is noted that whereas individual names have, for convenience, been applied to different parts of the deposit, as "Hilltop," "Catillas," "Pilar," etc., the whole may be regarded as a single continuous orebody of irregular form. From the top of Proaño Hill to the lowest part of the Pilar orebody the vertical extent of the deposit is approximately 850 feet.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

Both the stockwork and the enclosing rocks have three characteristics which are favorable to mining operations and which have had an important influence in securing low mining costs.

1. The rocks drill very easily, except in a few minor areas of more intense silicification.

2. The ground stands remarkably well and a comparatively small amount of timbering is required to support the workings.

3. The rocks are friable and readily break into comparatively small pieces under the impact of high explosives.

Hilltop Orebody

The form and extent of the Hilltop orebody are shown in Figures 1 and 2,A. The outstanding features of the orebody that affected the choice of a mining method were:

1. It was comparatively shallow.

2. Its greatest horizontal extent was at or near the surface.

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3. The silver content of the ore varied considerably in different parts of the deposit, so that many points of attack would be necessary to maintain a balanced production.

The shape and position of the orebody indicated the choice of an open-cut mining method. Steam shoveling and glory holing were considered, and the latter method was selected for the following reasons:

1. Glory-hole mining is extremely simple in detail, and could be carried out by the untrained labor available without assistance of the large foreign organization that would have been required by the steam-shovel method.

2. To secure comparable costs, large steam-shovel units would have been required, and the orebody could have been attacked in only a few places, whereas with the glory-hole method many points of attack would be available and a balanced production could be more easily maintained.

A reserve of broken ore could be maintained in the glory holes to assure a constant milling rate, whereas any serious breakdown in shovel operations would entail an immediate reduction in milling rate.

4. The preliminary expense for equipment and preparation was greatly in favor of the glory-hole method.

San Nicolas Orebody

The location and extent of the San Nicolas orebody will be apparent upon reference to Figure 1. It constitutes a westerly extension of the Hilltop orebody, lying along the principal fractures under the San Pedro fault, and in an area of intense cross fracturing.

Apart from the principal San Nicolas orebody, numerous smaller orebodies, consisting of flat-dipping lenses, extend in all directions, and add considerably to the total tonnage of ore. The area had been extensively worked by former operators, and a maze of old workings was accessible, extending downward to elevation 38.

The principal part of the orebody lay in a position favorable for the glory-hole method. The smaller and outlying extensions of the orebody were mined also by small glory holes from the surface, or by underground open stopes on the veins which were equivalent to glory holes insofar as little or no shoveling was required.

Catillas Orebory

The shape and extent of the Catillas orebody will be apparent upon reference to Figures 1 and 2,B. This orebody had the following principal features:

1. It had been extensively worked by former operators, and the mass of the orebody was badly shattered by the caving of a large proportion of the

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old workings. Many of the old stopes were filled with material which would constitute ore under present conditions.

2. The silver content of the orebody was somewhat higher than that of the Hilltop, but not sufficiently so to justify the use of a costly underground mining method.

3. The wall rock adjoining the orebody was relatively hard, and undisturbed by old workings.

4. Certain blocks of very good ore were hard and solid, and would require special attention to insure their clean extraction.

5. The orebody, as indicated in Figures 1 and 2,B, lay in two distinct parts, called the North and South branches, separated by a block of waste.

6. The 0-meter haulage way through which the Hilltop orebody was being mined passed through the upper part of the Catillas orebody. Production from the Catillas orebody, therefore, could not be started until the Hilltop was practically exhausted.

7. This orebody ranked next in size to the Hilltop, and would be required to produce the bulk of the daily tonnage when the latter was exhausted.

Nos. 6 and 7 were the deciding factors in the choice of a mining method. Glory-holing was the only method that would meet the requirements for daily tonnage. The selection of this method was, however, fortified by other considerations. An efficient glory-hole organization was available and the mining of this orebody would follow naturally upon the Hilltop operation. In spite of the waste stripping that would have to be done, experience in the Hilltop operations indicated that the glory-hole method would give a lower mining cost on the Catillas orebody than any other method. The block of waste between the two branches of the orebody presented a problem. If silver prices were high, this block would pay for extraction under a cheap method and dilution from it would not be a vital consideration. If silver prices were low, this block of waste would have to be separated, and this separation would be more easily accomplished under the glory-hole method than under any other. The shape of the orebody was peculiarly favorable to the glory-hole method. Although complete extraction of the orebody by glory-hole mining would not be possible because of the excessive stripping of waste that would be required, the bulk of the ore could be rapidly and cheaply extracted by this method, leaving the lower and richer portions of the orebody in a favorable condition for extraction by other methods, such as top slicing or block caving.

The Catillas and San Nicolas orebodies were mined concurrently, after the exhaustion of the Hilltop orebody.

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Pilar Orebody

The position, form, and extent of the Pilar orebody are shown in Figures 1 and 2,C. The orebody had the following outstanding characteristics:

1. It was long, narrow, and deep-seated.

2. Its silver content was nearly twice that of the Hilltop and Catillas orebodies.

3. A high-grade streak in the vein had been mined by former operators, and the old stopes had either caved or were filled with material now classed as ore. The payable ore was made up of these old fills, pillars left in the vein, and very irregular areas of good ore extending into the walls.

The governing factor in selecting a mining method for this orebody was its relatively high silver content, which made it necessary to mine it with a minimum of dilution. Glory holing was rejected because of the prohibitive amount of waste that would have to be stripped. Any overhand stoping method was out of the question, because the badly broken nature of the orebody would make the operation extremely hazardous, if it could be carried on at all. Any caving method appeared certain to result in heavy dilution in the irregular parts of the orebody, and might entail the entire loss of some of these extensions.

Top slicing was selected as being the cheapest method that would permit complete and clean extraction of the ore with minimum dilution. A new organization would have to be trained; but, on the other hand, the daily rate of production from the orebody was not a factor, because of other and ample sources of production.

San Pedro Orebody

The position, form, and extent of the San Pedro orebody are shown in Figures 1 and 2,A. This orebody represents an irregular extension of the Hilltop orebody at the intersection of the San Pedro fault with the main fractures of the Catillas system, in a zone characterized by pronounced transverse fracturing. Both orebody and walls were exceptionally firm and solid, though honey-combed by the open stopes of former operators. The orebody was irregular in form, and consisted of streaks of high-grade ore between masses of lower grade but payable material. While relatively unimportant in comparison with other orebodies, it contained an important tonnage of higher than average grade silver ore.

The orebody could not be completely mined without weakening the walls of the Catillas orebody and thus imperiling the glory-hole operation. If mined by the glory-hole method, as part of the Catillas operation, a heavy tonnage of waste would have to be stripped, which would have to be hoisted at a time when the general shaft was already taxed to capacity by the Catillas operation.

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In view of all conditions, the decision was reached to mine out the areas of better-grade ore by open-stope methods, which as finally developed amounted practically to underground glory holing. Ample pillars of lower-grade ore would be left, which could be recovered by caving methods after completion of the Catillas glory-hole operation.

METHODS OF SAMPLING AND ESTIMATION

The tonnage estimate of the Hilltop orebody, made by the Fresnillo Mining Co., was based on extensive sampling of quarry faces and floors, and of a few crosscuts driven by the company. Other parts of the orebody were sampled from the maze of old workings by drilling more than 700 holes with hand steel. These holes usually were 3 meters deep, and were drilled at right angles to the main fractures at any given point. In taking over the property the Mexican Corporation accepted the tonnage as stated, but reduced the estimated silver content by 10 per cent. The operating results when the extraction of the orebody was completed indicated that the true silver content of the orebody was about half-way between the two estimates.

The following general remarks relate to the orebodies other than the Hilltop that were developed by the Mexican Corporation.

The orebodies were developed on the several levels by parallel crosscuts at intervals of from 50 to 100 feet. These crosscuts were carefully channel-sampled in blocks 2 meters long. Assay maps were made for each level, and upon these maps the orebodies were delimited, giving consideration to the mining method to be used. For instance, sharp reentrant angles could be allowed where top slicing was to be used but were not permissible where the ore was to be mined by glory holing because of the difficulty of closely controlling the operation. In thus "smoothing" the outlines of the orebody some ore was excluded and some low-grade material was included at its assay value; the intention was to make an estimate of recoverable tonnage that would be checked by operating results.

Vertical and parallel cross sections at intervals of 5 meters were constructed from the assay plans. At each level an assay value was assigned to each cross section by weighting its relation to the two nearest crosscuts. An assay value for each ore area on the cross section was calculated by weighting the assay value of the upper and lower levels bounding it. Ore areas were then measured by planimeter on each of the cross sections, and a total assay value for each cross section was calculated by weighting its component parts.

Cubic contents were calculated by averaging the areas of each pair of adjacent cross sections and multiplying by the interval of 5 meters between them. The silver content of each of the 5-meter blocks was obtained by weighting the assay values of the two cross sections enclosing it.

The specific gravity of the ore was determined by experiment. The figure used varied from 2.2 to 2.5, depending on the proportion of broken ground, old fill, and solid material.

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In estimating silver content, a preliminary discount of 10 per cent was applied to actual assays, as allowance for probable errors in sampling and assaying. As an additional precaution, the work of the local sampling organization was checked by resampling a considerable number of crosscuts under the close and detailed supervision of an outside engineer.

In general the estimates of silver content have checked quite closely with operating results. Recovered tonnages have usually been somewhat higher than the estimates, probably because of the fact that wherever it was difficult to determine specific gravity accurately, because of the presence of old fills, the more conservative figure was used.

In laying out the Catillas glory holes and estimating the ore and waste contained in them, theoretical contour maps were drawn to represent the interior surface of the finished pit. The orebody had its greatest horizontal extent at about the 44-meter level. The first contour drawn was therefore laid out at this elevation to enclose the entire orebody within a smooth outline containing as little waste as possible. From the 44-meter level to the surface, contours were constructed at 5-meter vertical intervals, the slope in each part of the glory hole being governed by the character of the ground. In the parts of the glory hole where the ground was firm and solid a slope of 2 to 1, or 63-1/2° was used. According to experience in mining the Hilltop orebody this was the maximum safe slope for solid ground. Its use in the Catillas orebody, which is now nearly mined out, has been justified by the results. Where the ground was broken or soft a slope of 45° was used. From the 44-meter level downward the governing factor in the shape of the pit was the location of the several chutes and grizzlies at the 105-meter level. Contours from the 44-meter level downward were also constructed at 5-meter vertical intervals, with maximum slopes of 63-1/2° to the bottom of the theoretical pit. These contours were then transferred to the cross sections, and the theoretical slopes were constructed. Each cross section was then studied in detail as to (a) ore left outside the glory hole, and (b) waste-to-ore ratio of the ground at the periphery of the glory hole. In considering the inclusion or exclusion of any given section of ore by modifying the shape of the pit, the amount of waste affected by the change, or the waste-to-ore ratio, was considered in each case, as well as the grade of the ore. The higher the silver content of the ore, the higher would be the economic limit of the waste-to-ore ratio. Ore on the periphery of the pit was considered in detail with this viewpoint, and the final outline was arrived at by the cut-and-try method. In parts of the pit where the orebody was irregular in both plan and section the contours had to be revised numerous times before the final design was determined. Falling silver prices in recent years have made it necessary to revise the pit design several times to meet the new conditions.

Waste tonnage was determined in the same manner as the tonnage of ore.

Figure 3 is a typical cross section of the Catillas orebody, showing the relation of the estimated and actual pit lines to the orebody itself.

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METHODS OF DEVELOPMENT AND STOPING

General Development

Hilltop Orebody

The outlines of the Hilltop orebody were closely delimited by the maze of quarries in Proaño Hill, and by crosscuts driven from them at an elevation of 25 meters above the 0 level.

Figure 4 shows the general plan of the haulage way driven at the 0-meter elevation to permit large-scale mining of the deposit by the glory-hole method. The bottom of the orebody was from 20 to 60 feet above the haulage-way, except for the two downward extensions represented by the Catillas and San Pedro orebodies, so that practically all of the haulage way was driven in waste. All of the Hilltop orebody was extracted through this haulage way, as well as a part of the Catillas. Ample room for a mill site remained on the hillside below the portal of the haulage way.

The standard cross section of the haulage way was 10 feet wide by 9 feet 6 inches high. Timber was required in only a few places. All curves were of 100-foot radius. The track was of 50-pound rails laid to a 36-inch gage on 6 by 8 inch by 6-foot pine ties at intervals of 2 feet 6 inches; it was well ballasted. The grade was 0.5 per cent in favor of the load.

Crosscuts were driven to prospect several favorable blocks of ground. These crosscuts were 3 feet 6 inches wide and 6 feet high, and were driven without track. Wheelbarrows were used to handle the material to the main haulage ways, where it was shoveled into cars. This work was done very cheaply.

Glory-hole loading stations were installed at intervals of 150 feet along the main haulage ways, staggered so as to be on the corners of equilateral triangles. Figure 5 illustrates the standard loading station installation.[4] Raises of circular cross section 10 feet in diameter were driven from the grizzly chambers to the surface. Advantage was taken of old workings in planning these raises; they were driven at various slopes and directions to reach their objectives with the least possible amount of work. The raises ranged from 125 to 200 feet in length.

Catillas and Pilar Orebodies

Coincident with preparations for mining the Hilltop orebody, unwatering of the deep parts of the old mine was begun through an old shaft to the east of the Pilar orebody. From this shaft access was had to the Pilar and Catillas areas through an old level at an elevation of from 105 to 112 meters. The old workings were useless for any modern mining purpose, as they were

4 - Baker, T. C., Fresnillo Glory-hole Mining Practice: Eng. and Min. Jour.-Press, vol. 116, Dec. 1, 1923, pp. 931-942.

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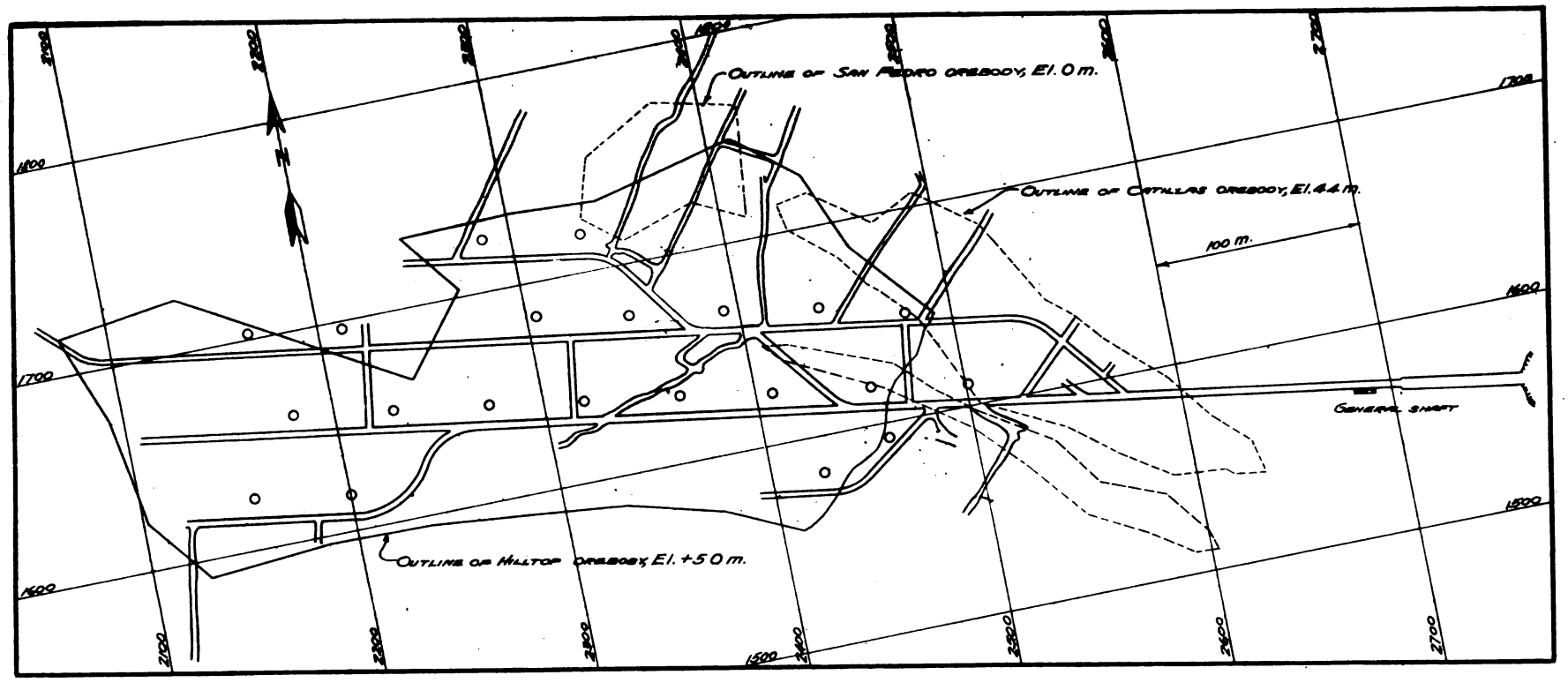

Figure 4.- General plan of haulage way at 0-meter level

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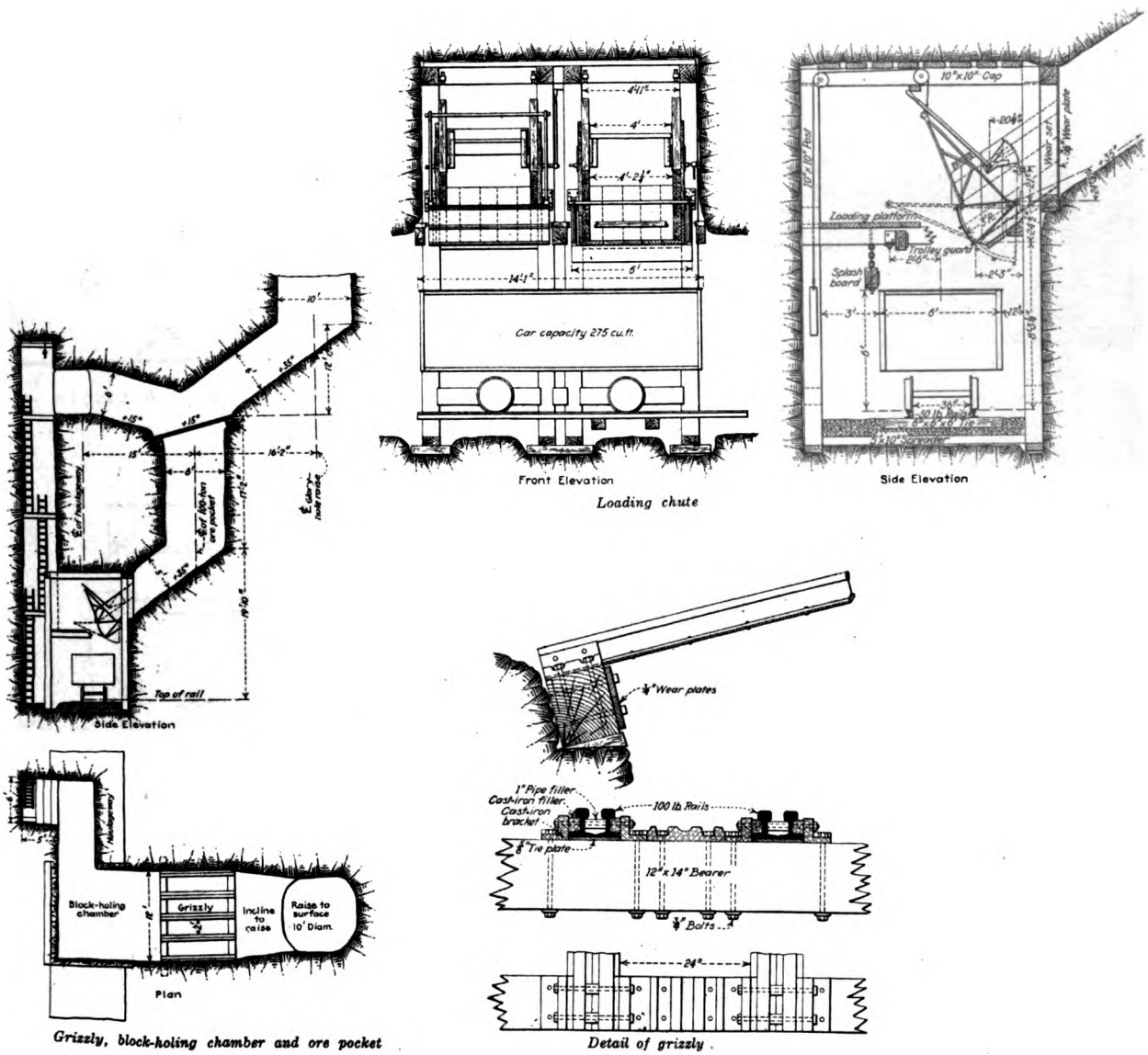

Figure 5.- Standard loading station installation, showing grizzly chamber, ore pocket, loading chute, and details of grizzly construction

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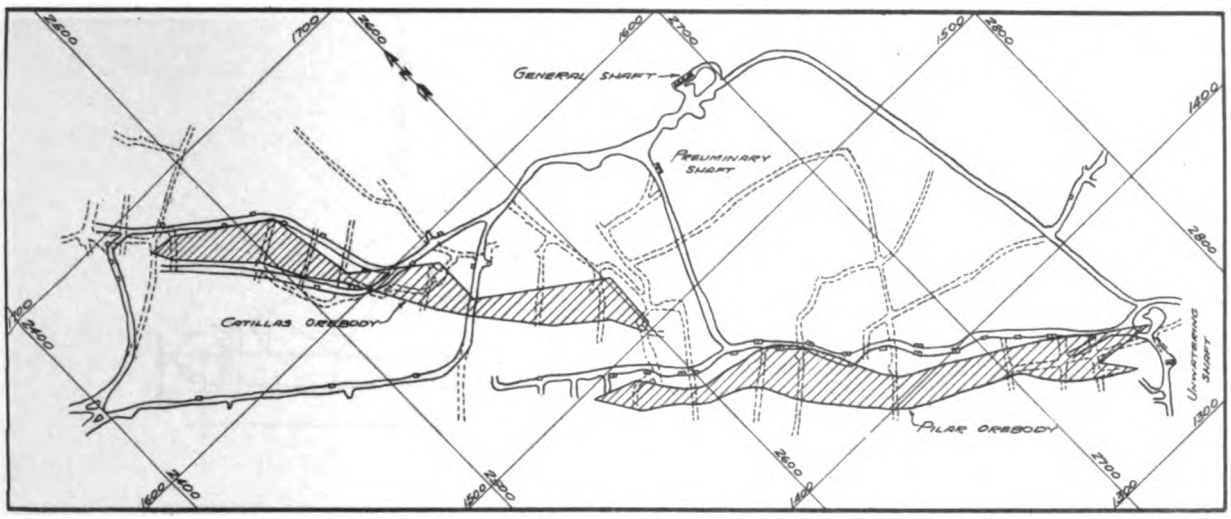

Figure 6.- Part of 105-meter level. Old workings and preliminary crosscuts shown in dotted lines

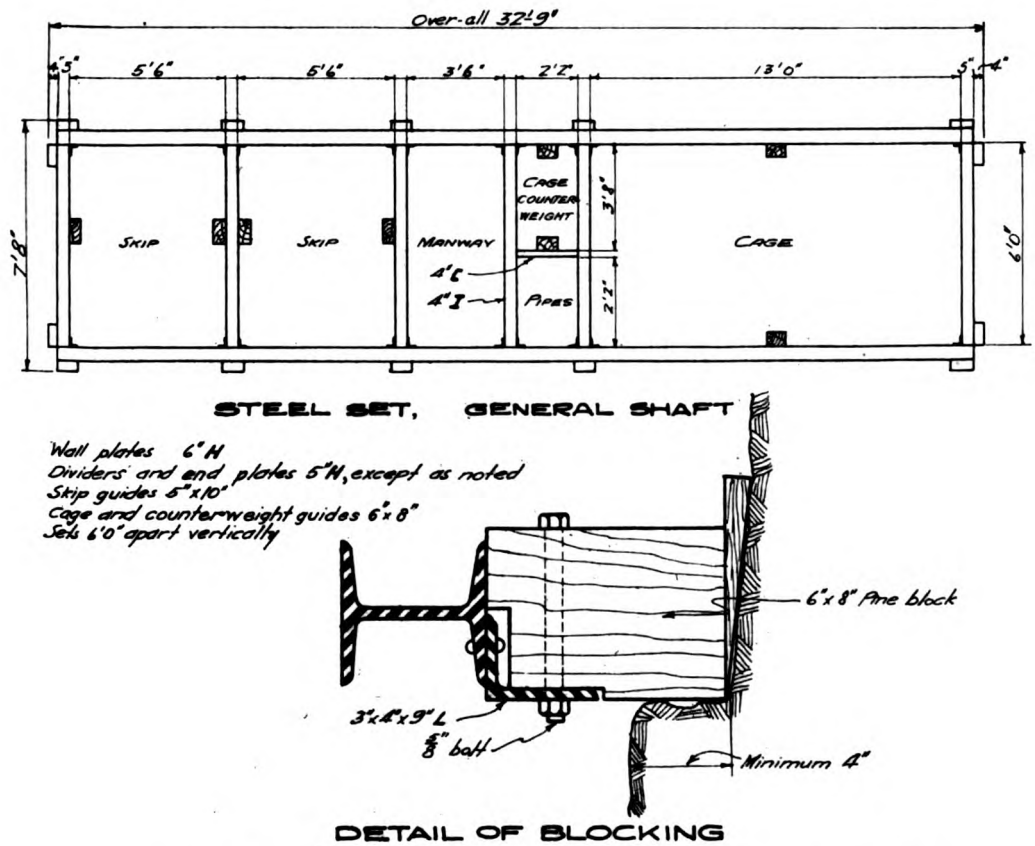

Figure 7.- Steel set and method of blocking, general shaft

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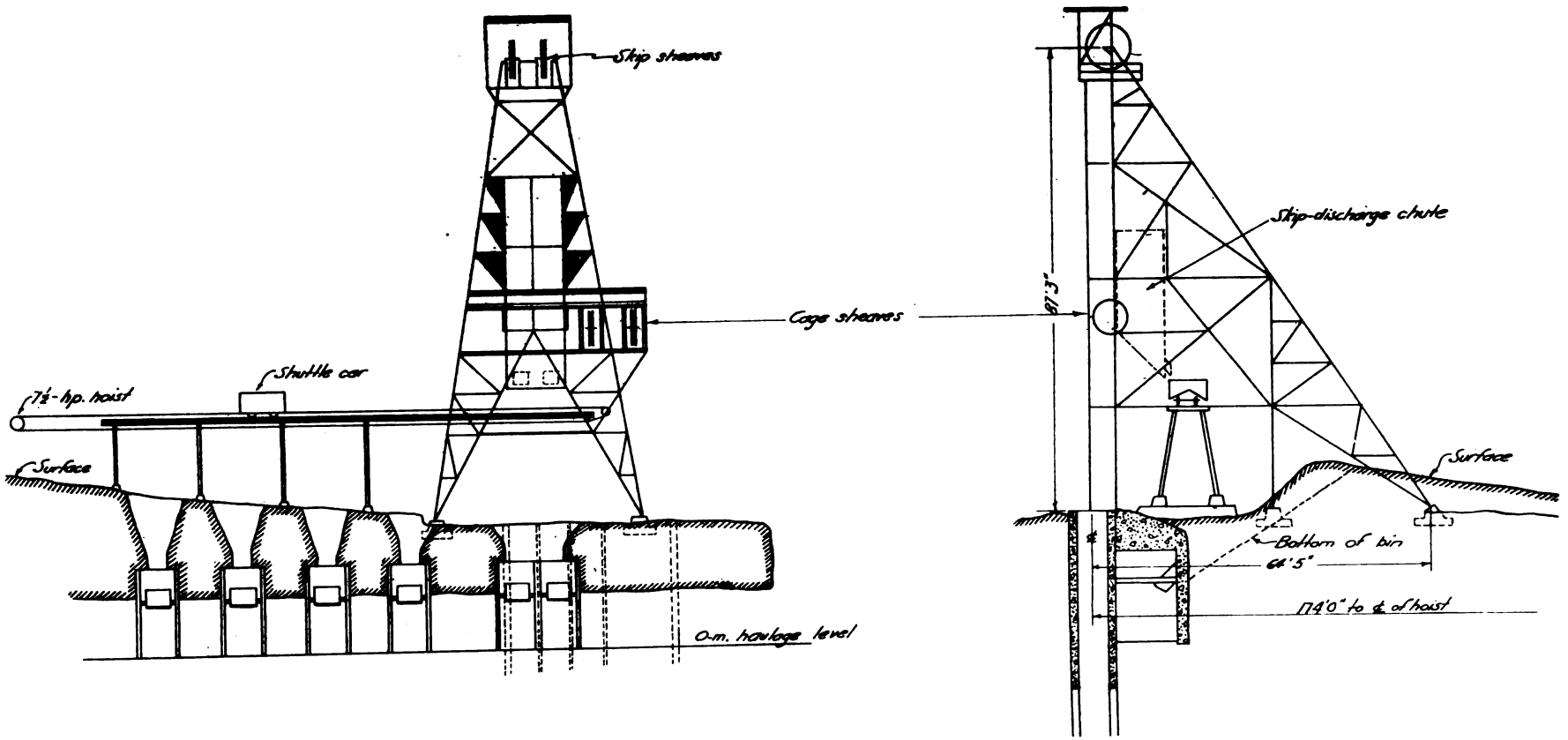

Figure 8.- Front and side elevations at General shaft, showing headframe and arrangement of loading bins

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crooked and drifts varied in elevation as much as 10 meters between ends. However, they gave immediate access to the desired parts of the mine, and made it possible to determine the outlines of the orebodies very quickly and cheaply. Three by six foot crosscuts were driven from these old workings at advantageous points, the material from the crosscuts being stored in the old workings. Figure 6 is a plan of the 105-meter level, and shows a part of these old workings and the crosscuts driven from them.

With the outlines of these two orebodies established and a sufficient tonnage of ore indicated, comprehensive development was undertaken. An old shaft, extending to the 108-meter level and conveniently located, was equipped for temporary service, and levels were driven from it at elevations of 44 and 105 meters. These two levels were placed to take advantage of certain old drifts that were in good condition and at suitable elevations to be used. In consequence, these levels were opened up very rapidly.

The 44-meter level was driven primarily for the purpose of development rather than extraction, and the Catillas and Pilar orebodies were thoroughly explored at this horizon. Drifts and crosscuts were 5 by 7 feet in cross section. In and near the orebodies the workings had to be timbered. Subsequently a part of the level was enlarged for haulage to 7 feet 4 inches high above the rail, 5 feet 6 inches wide at the top, and 7 feet 4 inches wide at the bottom inside of the timbers. Caps were of 10 by 10 inch timber and 7 feet long. Posts were of 10-inch round timber 8 feet long.

The 105-meter level, part of which is shown in Figure 6, was established as the principal haulage level, and the main drifts were placed to skirt the orebodies. These workings were of the standard haulage drift cross section described above.

As unwatering of the old mine continued, access was gained to the Pilar area through old levels at elevations 142 and 165, and additional crosscutting was done from these old workings. The Pilar orebody was proved to extend downward to within a few meters of elevation 165, and the lower limit of the Catillas orebody was determined to be at approximately elevation 135.

A haulage level was driven from the General shaft at elevation 165. This level had to be timbered in and near the orebodies, and was driven with the standard cross section.

General Shaft.--The General shaft was sunk for mining the Catillas and Pilar orebodies. It was located alongside of the 0-meter haulage way so that it would be served by existing tracks. The collar of the shaft was placed at a suitable elevation to provide ample storage in the surface bins above the 0-meter haulage way at the minimum cost. The shaft is 7 feet 8 inches wide by 32 feet 9 inches long and is lined with steel sets; a horizontal cross section of the shaft and a detail of the method of blocking the steel sets are shown in Figure 7. The general surface arrangement of the shaft and ore bins is shown in Figure 8.

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The shaft was sunk from the surface in full cross section. At the same time raises of small cross section were driven from the 44-meter and 105-meter levels. After the raises were connected this section of the shaft was stripped and the steel placed from the top downward, the waste from stripping being drawn off through the raises. The method was cheap and safe. Below the 105-meter level the shaft was sunk and lined in full section.

The shaft is in very good ground and is largely without lagging. Gunite was used in some places where the ground had a slight tendency to slack and scale. Masonry was used to support a number of large slabs that developed after shaft timbering was complete. In several places where lagging was required, worn rods from the Marcy mills were used. A hook was forged on one end of the rod to fit over the outside flange of the steel wall plate. The rods hang vertically and the lower ends engage the next set below. They are spaced from 10 to 20 inches apart, depending on the thickness of the rod and the weight to be supported. Sheets of corrugated iron were placed behind the rods with the corrugations horizontal, and the space between the sheets and the ground was tightly filled with waste rock. The corrugated sheets were painted on the back side with heavy crude oil to retard rust. Installations of this type made in 1923 have required no subsequent attention.

Ladders are of steel and manway landings are of concrete. The division between the manway and skip compartments, above the 135-meter level pockets, is of 3/4-inch mesh steel screen. Below these pockets, the division is of 2 by 10 inch planks, fitted tightly in the channels of the 5-inch H-beam divider. A tight division is necessary to confine the loading spill to the skip compartments. This timbering is the only fire hazard in the shaft, but no satisfactory substitute for it has been found within reasonable cost limits. The wooden lagging has the additional great advantage that it can be replaced safely from the manway compartment without stopping the skips. As the shaft upcasts warm, moist air, this lagging has been damp at all times, and represents only a small fire risk. With the exception of that at the 105-meter level, all the shaft stations are in hard ground and are untimbered.

The shaft is concreted outside of the steel sets for 30 feet below the collar. The headframe is of steel. Ample storage capacity is provided in the bins at the collar, at a very moderate cost.

The cost per foot of excavating and "timbering" the General shaft from the collar to a depth of 800 feet was as follows:

Cost per foot (U. S. currency)		
Excavation	$41.94	
Timbering (steel sets)	116.61	
Tramming and hoisting	11.10	
Explosives	9.50	
Miscellaneous	14.93	$194.08
Labor	90.18	
Supplies	99.02	
Power	4.88	$194.08

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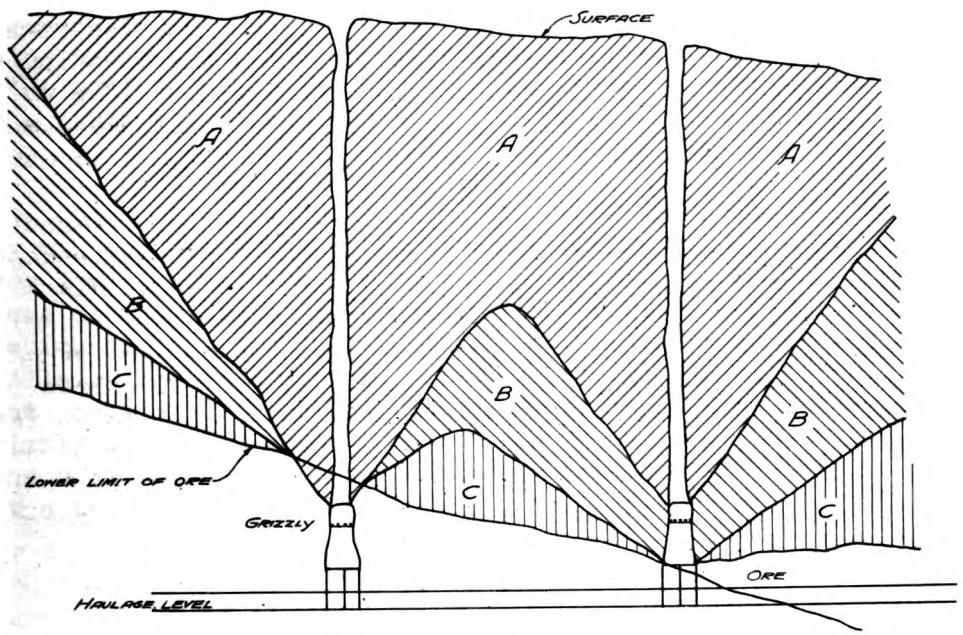

Figure 9.- Typical section of glory hole: A, First stage; B, second stage; C, final stage

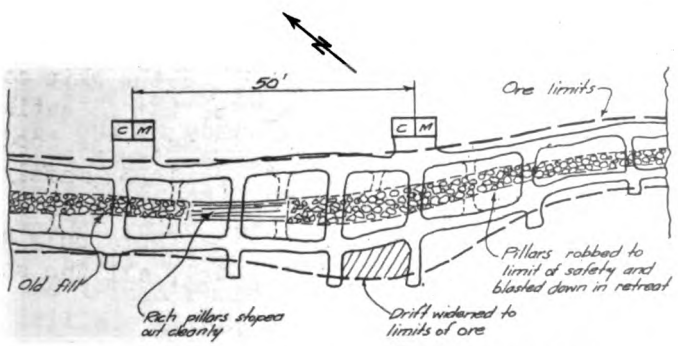

Figure 10.- Typical caving sublevel, Pilar orebody, elevations 30, 37, 44, 54

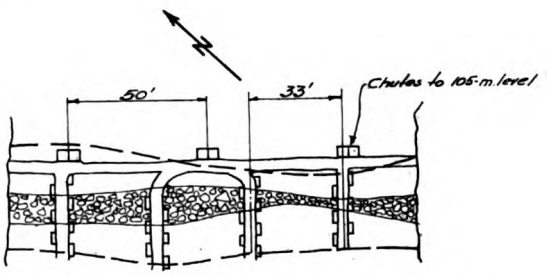

Figure 11.- Typical nano-tramming caving level, 65-meter level, Pilar orebody. Crosscuts solidly timbered, with drawing chutes on both sides in alternate sets

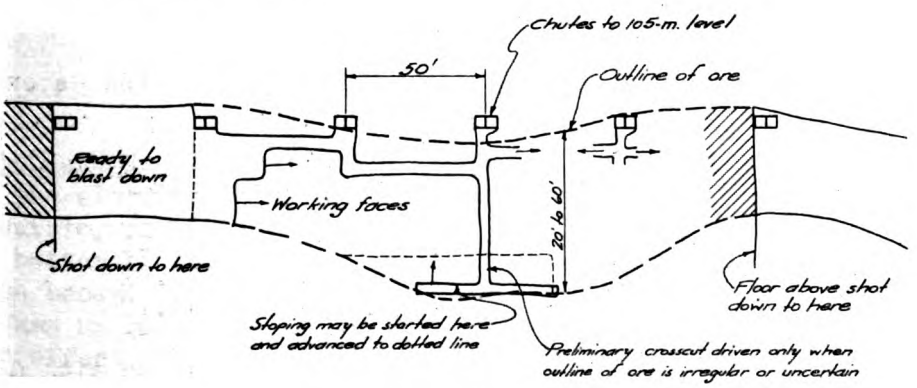
Chutes to 105-m. level
50'
Outline of ore
Ready to blast down
Working faces
20 to 60'
Shot down to here
Floor above shot down to here
Stoping may be started here and advanced to dotted line
Preliminary crosscut driven only when outline of ore is irregular or uncertain

Figure 12.- Typical top-slice floor

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The above figures include all items chargeable against the shaft, calculated as cost per vertical foot of finished shaft. The cost of the concrete collar, the loading pockets for the 105-meter and 165-meter levels, and the construction of the surface bins, is included. The cost of the headframe, hoist, and other equipment is not included.

San Nicolas Orebody

This orebody was developed by sublevels at elevations 0 and plus 15. Ore and waste from development were stored in old workings. A drift on the 105-meter level was extended to a point underneath the orebody, and an ore pass was driven which connected to old workings at elevation 38, where a grizzly was installed. Branches from this ore pass above elevation 38 reached the most important parts of the ore body so that the ore, when broken, ran direct to the grizzly without shoveling or tramming. Ore from parts of the orebody beyond the economic reach of branch raises was trammed on the 0-meter level to one of the branch raises.

Development Details

The adoption and enforcement of standard drilling and blasting practice has not been practicable at Fresnillo, principally because of widely varying ground conditions. Rock drills have been extensively tested and the most efficient models have been supplied. Much drifting and crosscutting have been done with unmounted jackhammer drills weighing 55 pounds; mounted machines have been used where the ground was harder or greater speed was desired, especially in the larger heading. Raising has been done with stoper drills, and in some instances with mounted machines. Drill bits are of the standard double-taper pattern. Three kinds of hollow drill steel are used: 7/8-inch hexagonal for the jackhammers, 1-inch round for the stopers, and 1-1/4-inch round for the mounted machines. The first mentioned has been used in the glory holes; in development headings jackhammers have been equipped to use the 1-inch round steel. All drilling in development headings is done with wet drills. Either 30 or 40 per cent strength gelatin dynamite is used in development work. Mucking is done by hand, using a D-handled, round-pointed shovel.

Stoping

Hilltop Orebody

Glory-Hole Method.--Figure 9 is a typical cross section through the Hilltop glory holes. The slope in the glory holes was maintained at 55° or steeper as long as possible, or until the collar of the glory hole had been mined down either to the bottom of the orebody or as close to the grizzly as possible and a flatter slope became imperative. When the slope is steep the ore is more thoroughly broken up by falling after the blast and grizzly expense is greatly reduced. This effect is most notable, of course, when working high up on the sides of the slope. The first cut in each cycle was taken at the bottom of the glory hole around the collar of the raise. Successive benches 8 to 10 feet in height were then mined; the direction of retreat was upward toward the rim. When working in loose ground this procedure was occasionally reversed from

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considerations of safety. When the bench had been completed to the rim, all sides of the glory hole were carefully barred down while the glory-hole raise was being emptied, so that the collar of the raise could safely be reached to start a new cycle. It was advantageous to maintain the steep slope as long as possible, to avoid the necessity of working the broken ore into the glory hole by hand. In cases where the bottom of the orebody was at a distance above the grizzly it was profitable to start the cyle by taking a cut in waste around the collar of the glory hole. This waste was handled as such and not mixed with the ore, and the procedure was followed until the cost of breaking and handling the waste overbalanced the cost of mucking in the glory holes. Thereafter, slopes in the glory holes were successively flatter with each cycle, until a minimum of approximately 35° was reached. With slopes flatter than 35° it became impossible to mill down the ore without hand mucking. To extract the ore remaining between glory-hole raises, a horizontal bench was started at the bottom of the ore and wheelbarrows were used. Although scrapers have not been used in the glory holes at Fresnillo, the writer believes that under favorable conditions, and particularly in irregular orebodies, large scrapers could be used very successfully with glory-hole mining. Raising would be reduced and the mining of some waste to create safe slopes might be avoided.

Where the ore extended below the grizzly the grizzly was finally removed and the glory-hole cycles were started at successively lower levels--in some cases until the chute mouth was reached. During this stage as much block holing as possible was done in the glory holes to avoid excessive blasting in the chute mouths with consequent loss of time in loading and tramming. Production from the glory hole was relatively small during this final stage, but this was no great disadvantage, as the loss of production was easily made up from other glory holes. In a few places the tonnage remaining between glory holes was sufficient to justify the installation of intermediate loading stations and short raises without grizzlies.

Waste stripping in the Hilltop orebody was small in amount, and consisted mainly of the separation of blocks of waste surrounded by ore, plus a small amount of material from the rims of the holes that had to be mined as a safety precaution.

The safe limit of glory-hole slope is a factor that varies greatly in different localities. At Fresnillo slopes in excess of 70° have been carried with entire safety, where they did not exceed 100 feet in height. The steeper slopes were inspected carefully every day. Sloughing occasionally took place, but it was small in amount and always gave ample warning. In planning glory holes where waste stripping was a factor and the ground solid, a slope of 63-1/2° was used; where the wall rock was less favorable, slopes laid out as flat as 45° were used.

Drilling in the glory holes was done with 55-pound jackhammers, using 7/8-inch hollow-hexagonal steel, the light weight of which was an advantage because of the distance it had to be carried by hand. The machines were fitted with powerful blowing attachments for cleaning out the holes. The

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pressure at the drills was a minimum of 75 pounds. Drilling was done without water. Water lines in the glory holes would have been an extra item of cost, and upkeep on the dry machines was less than if water had been used. Wet drilling would have made the slopes slippery and dangerous in many places, and it was considered that the dust had no harmful physical effect inasmuch as drilling was done in the open air.

Holes averaged about 9 feet in depth, and were given a burden of between 3 and 4 feet. The average machine performance was about 120 feet of hole per 8-hour shift. All drilling was done on contract on a basis of footage drilled, the rate per foot ranging from $0.028 to $0.036, U. S. currency. The drill crew consisted of a driller, a helper, and a tool nipper, the latter serving two machines. A contractor usually had two machines, or more, and operated one of them himself. Though paid on the basis of footage drilled, he was required to bar down the benches, except in special cases, and to carry his own steel. Foot trails were maintained to all working faces, and steel was carried by the tool nipper from a central magazine on the rim of the glory holes to the machine. All equipment and steel were carried out of the glory holes at the end of the shift and checked in at the magazine to prevent loss. Drillers worked both day and night shifts, the work at night being confined to places of lesser hazard. The average production per machine-shift was over 130 tons for the whole Hilltop operation.

The drilling practice described above was suited to the nature of the ground and the quality of labor available, and it is thought to have given about the best results possible under such conditions. While it was satisfactory in operation and gave excellent results, the contract system used had a serious defect: the contractor was interested only in drilling the hole as quickly and as easily as possible, and its location was a matter of no interest to him. Unless supervision was close and careful, many holes of too great or too little burden would be drilled. At Fresnillo, a "hard-boiled" policy of refusing payment for such holes was adopted, but the point was never reached where the tendency was entirely corrected.

Blasting was done in daylight only, by a special crew of men under a boss of long experience and proved careful habits. The men worked on company account and were obliged to adjust their hours to the demands of the work at different times. Holes were sometimes fired at noon, sometimes at the end of the day shift, and sometimes early in the morning before the day shift came on. Usually, the blasters came on with the day shift and made up the estimated number of primers for the day's work. The primer consisted of a No. 8 cap, one stick of explosive, and a fuse from 6 to 9 feet long. During the day shift, holes were loaded in places where drilling was not going on, and were occasionally blasted at noon. Loading of the holes drilled on day shift began about the time the drillers were finishing their work, and the shots were fired about 40 minutes after the drillers had gone. The explosive used was bulky, with a comparatively low velocity of detonation, and was rated at about 35 per cent strength. It was selected after extensive tests of a number of brands on the basis of tons of ore broken per pound of explosive.

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The average charge was about six sticks of explosive to the hole. The primer was placed near the middle of the charge, which was well tamped; the top of the hole was filled with the dry drill cuttings for stemming. The ends of the fuses were assembled in groups of three or four, as far as possible, and the firing was all done by hand, using a short piece of nicked fuse as a spitter. An air whistle was used to give warning of the blast. The work of spitting the holes was divided carefully among the men, each of whom would spit his allotted holes and retreat to the rim of the glory holes over an appointed route. There was never an accident to a blaster in this kind of work.

A grizzly opening of 24 inches was used throughout in mining the Hilltop orebody. This wide opening was made possibly by use of large chute doors and 10-ton tipple-dump ore cars on the 0-meter haulage level which delivered directly to the gyratory crusher at the mill.

When working high up in the glory holes, and while the slopes were steep, the ore was well broken up by falling, and the amount of block holing required on the grizzlies was comparatively small. Grizzly expense increased very materially when working around the collars of the glory holes and when the slopes were flat. Grizzly men worked in pairs, each pair tending from one to three grizzlies, depending on the coarseness of the ore and the demands of production. Point bars and 8-pound hammers were used to work the ore through the grizzly until it became clogged with large rocks, which would then be drilled and blasted. A 21-pound jackhammer was employed for block-holing, using 7/8-inch hexagonal steel; drilling was without water. Most of the holes drilled were less than 12 inches deep. The men retreated to the haulage level below when the shots were fired. Both the bulk explosive previously mentioned and 30 per cent strength gelatin dynamite were used in this work. Only 4 per cent of the total explosive consumed in mining this orebody was charged to secondary blasting on the grizzlies.

The glory-hole raises had an original diameter of 10 feet, and soon wore out much larger. They never "hung-up" except in rainy weather, and not then if worked continuously so that the ore was kept moving and not permitted to pack.

The brow of rock over the grizzly on the side next to the glory hole was found to wear severely, so that eventually the grizzly would become completely flooded with ore and be difficult to work. It was necessary in such cases to install a heavy baffle to hold back the ore in the glory-hole raise. Steel beams were used for baffles in some instances, in others the baffle was made of four 12 by 12 inch timbers, reinforced on the wear side with 1/2-inch steel plate and faced with 50-pound rail riveted to the plate in a vertical position. The baffles usually were set in hitches cut in the rock; in some cases where the raise had worn wide as well as high it was necessary to line the sides of the hole with masonry to reduce the opening to its original width so that the baffle would not have too great a span.

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Under favorable conditions, 1,650 tons of ore has been put through a single grizzly in an 8-hour shift. In excess of 22,000 tons of ore has been passed through a single grizzly in a month, working three shifts a day. It was never possible to work a grizzly at full capacity for an entire month because it was never desirable to concentrate production in a single small area for a period of that length. In the Hilltop orebody the absolute capacity of a glory hole, when slopes were steep and conditions favorable, would have been in the neighborhood of 35,000 tons per month. In actual practice a monthly production of 80,000 tons was spread over about 10 glory holes in order to maintain a silver content in the ore produced that would correspond to the average grade of the orebody. The average rate of production in the Hilltop ore body was between 8,000 and 10,000 tons per grizzly per month.

The effect of rainfall has to be considered in planning any open-cut mining operation. Rainfall at Fresnillo, though slight, has been of considerable importance at times. It amounts to from 10 to 12 inches annually, and occurs as heavy showers of short duration, concentrated in the last six months of the year. The time lost by ore-breaking crews is negligible, but during rainy weather work in the pits is confined as much as possible to the upper parts, as the moisture tends to loosen masses of rock which are normally unstable. The most serious effects of rain on the mining operations were the tendency of wet ore to pack in the raises, and the flooding of underground workings with mud and water from the pit drainage. The former difficulty was met by keeping the ore moving in the raises, or by drawing them nearly empty, as stated above. This sometimes necessitated blasting twice a day or oftener. The flooding of the tracks on the haulage often stopped production for short periods. The water could be drained off into lower workings, but the mud had to be shoveled into cars. However, tramming on a reduced scale always could be resumed after a few hours, and only rarely was more than 24 hours needed to complete the clean-up.

The most serious effect of rainfall on the Fresnillo operation as a whole was at the mill, where the capacity of the crushing plant sometimes fell off as much as 60 per cent when the ore was very wet.

Catillas Orebody

Glory-Hole Method.--Glory-hole loading stations for the Catillas orebody were installed on both the 44 and 105 meter levels, at irregular intervals, the irregularity being due to the broken character of the orebody and the consequent necessity of selecting favorable places even at the sacrifice of the desired interval. The standard loading station set is an exact replica on a smaller scale of the set used on the 0-meter level, except that single instead of double chutes were used.

Figure 3 is a typical cross section of the orebody, showing the ore outlines, estimated and actual outlines of the glory hole, and the arrangement of the glory-hole raises. A string of raises was put up from the 44-meter level along the north side of the orebody to facilitate separation of

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the V-shaped body of waste that lay partially above the ore. These raises were all provided with grizzlies about 30 feet above the 44-meter level, similar in design to those used in the Hilltop orebody. Long raises were driven from the 105 to the 44 meter level, taking advantage of old workings wherever possible, and continued through to the surface. Grizzlies were first installed about 30 feet above the 44-meter level, and as the operation progressed, were moved downward and installed successively at the 44, 75, and 95 meter elevations. Because of the sticky nature of some of the ore it was impossible to operate with high raises above the grizzlies, as such raises constantly hung up and gave trouble. The space in the raise below the grizzly was seldom filled, and never in rainy weather; consequently, this part of the raise never hung up.

The method of installing the grizzlies varied much in this orebody. Above the 44-meter level several of the raises to the surface required solid cribbing, and grizzly stations were constructed of heavy timber adequately protected with steel wear plates. In some of the raises masonry ore pockets 10 feet square inside and 30 feet high were built up from the 105-meter level to carry the grizzlies at elevation 95. However, all of the principal features of the grizzly design used in the Hilltop orebody were retained in the Catillas operation, as experience had proved that they gave the most satisfactory operating results.

The primary drilling and blasting practice followed substantially that developed in the Hilltop operation. Because of the broken condition of the orebody, the footage of holes drilled per machine-shift was notably less than in the Hilltop orebody, and the labor of barring down was correspondingly greater. Accurate figures as to tons per machine-shift are not available, but probably would be from 10 to 15 per cent under the Hilltop figure, which was about 130 tons. The favorable shape of the orebody, as seen in Figure 3, allowed steeper slopes to be carried throughout the operation. The orebody has been mined down practically to the 85-meter elevation, with only an insignificant amount of hand mucking. Greater precautions for safety had to be taken than in the Hilltop operation, and the slopes were constantly and carefully inspected for loose rocks and cracks. The glory-hole operation has been continued until the walls of the pit in the highest part at the northwest end are about 450 feet high, vertically, and are standing at an average slope of more than 60°. There has never been a serious accident due to a fall of rock from the walls, and only a few accidents that occasioned a loss of time for the injured men.

A grizzly opening of 16 inches was adopted for the Catillas orebody. A smaller opening was made necessary by smaller tramming equipment and smaller chute gates on the 105-meter level, and by the fact that the ore had to pass through loading hoppers and be hoisted in 5-ton skips in the General shaft. Because of the smaller grizzly opening, secondary blasting was a somewhat more costly operation than it had been in the Hilltop operation. Production per man-shift was lower, powder consumption on the grizzlies was higher, and the items of grizzly construction and maintenance were more costly. The foregoing were the principal factors accounting for the increased cost of ore-breaking in the Catillas operation as compared with the Hilltop.

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Pilar Orebody

Former operators had stoped a narrow high-grade streak in the Pilar orebody continuously through to the surface, and these old stopes for the most part had been left filled. From the surface to a short distance above the 44-meter level the orebody was too low-grade to mine. Careful consideration was given to the breaking down of this overburden and to the formation of a cushion of broken ground under which top slicing could be started with safety.

Caving Methods.--Exploration of the upper parts of the orebody indicated that its extent and silver content would not justify top slicing above elevation 89. Caving methods were therefore adopted to recover the ore above this elevation, with the further idea that this operation would adequately prepare the way for top slicing below the 89-meter level by thoroughly breaking the overburden and leaving it in a condition to follow the mat down. At the 44-meter level the orebody was 15 feet wide. Downward the width increased to 35 feet at the 65-meter level. Because of this narrow width, caving levels were driven at short vertical intervals. There were four caving levels, at elevations 30, 37, 44, and 54. (See fig. 10.)

Figure 10 illustrates the procedure followed on these four upper intermediate levels. A drift was driven in the comparatively solid north wall of the orebody, from which crosscuts were driven through the old fill and connected to each other on the south side at the edge of the ore. Where they penetrated old fill, the workings required timbering, and at times had to be spiled. Most of the work, however, was safely accomplished without timber. Old pillars of rich ore in the vein where found were mined out as cleanly as possible. The pillars between the crosscuts were then robbed to the limit of safety and blasted down in retreat. Sometimes the backs of the undercut openings were drilled and blasted, but not as a rule, for the ground usually caved readily when the supporting pillars were blasted out. The ore broke up well, only occasional large blocks requiring blasting in the chute mouths. The work on these sublevels resulted in the clean recovery of about 15 per cent of the better part of the ore, and left the remainder in a condition much resembling a completed shrinkage stope approximately 80 feet high and 300 feet long. The walls of the orebody, while not notably strong, were sufficiently so to prevent excessive dilution while the broken ore was being drawn.

The orebody was narrow, and would justify only one haulage drift and one line of raises from the 105-meter level. The drift was driven to skirt the north edge of the orebody as nearly as possible, and 2-compartment, timbered raises to the 65-meter level were located along it at intervals of 50 feet.

The 65-meter level was used as a hand-tramming level for the caving operation, and had its own independent system of raises. Figure 11 illustrates the method of caving and drawing off the ore at the 65-meter level, which followed in a general way the method then in use by the Miami Copper Co. Chutes opposite each other in contiguous crosscuts were connected by 30° inclined raises driven in the comparatively solid ground outside of the old

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fill. These raises were then widened to the limit of safety on the side away from the fill. Pillars between the raises and the old fill were next robbed as far as safety would permit and then blasted down. Drawing from the chutes was then started and carried on as evenly as possible over the entire length of the orebody, so as to prevent funneling. Very little timber was used in the undercutting operation.

Subsidence of the surface followed within a very few weeks after drawing off at the 65-meter level was started, first in a narrow zone closely following the old filled stopes, but soon extending laterally as the walls surrounding the caved ore collapsed. The tonnage and silver content of the ore drawn at the 65-meter level closely approximated expectations. As the cost per ton of ore drawn was quite low the success of the caving operation was indicated.

Following this drawing operation the orebody was undercut and caved at the 79-meter level by methods very similar to those used on the upper sublevels. However, the greater width and higher silver content of the orebody at this elevation required that the operation be carried out more carefully in detail, and care was taken to reach the limits of the ore in all directions and to undercut it thoroughly. No ore was drawn off at this level.

At the 89-meter level, a complete floor was taken out of the orebody by the square-set method, and again care was taken to reach the ore limits in all directions. The square-sets were blasted down in blocks as large as possible, to get the maximum shattering effect on the ore above.

At the 91-meter level, a combination of top slicing and brow caving methods was used, the details of which are not of special interest. A complete floor was taken off by this method, and all of the broken ore below the 65-meter level was drawn off. A mat of double 3 by 12 inch plank was laid

Top Slicing.--Top slicing started at elevation 94, and was carried down to elevation 162, at a point 10 feet above the 165-meter level, where the orebody terminated. The method used varied greatly in detail as new ideas were developed from time to time. Figure 12, however, illustrates the general practice. Figure 13 shows the usual method of timbering.

The work was done on company account at the start, but was later put on a contract basis. A unique feature was the system of payment. A prime necessity when using the top-slice method is to get the floor laid as soon as possible. To insure carefulness in this respect, the contract price for ore breaking was fixed at so much for each floor board of standard size that was laid. This price was carefully worked out and was never changed throughout the whole operation. Other payments were made for spiling, which was occasionally necessary, reinforcing of timber, etc., but more than 90 per cent of the money paid the contractor was calculated on the basis of the floor boards laid. Even the driving of advance drifts was put on this basis. After the adoption of this system, cases were very rare where the slice caved before the floor was laid. Moreover, it was to the contractor's pecuniary advantage to lay the floor as closely and tightly as possible.

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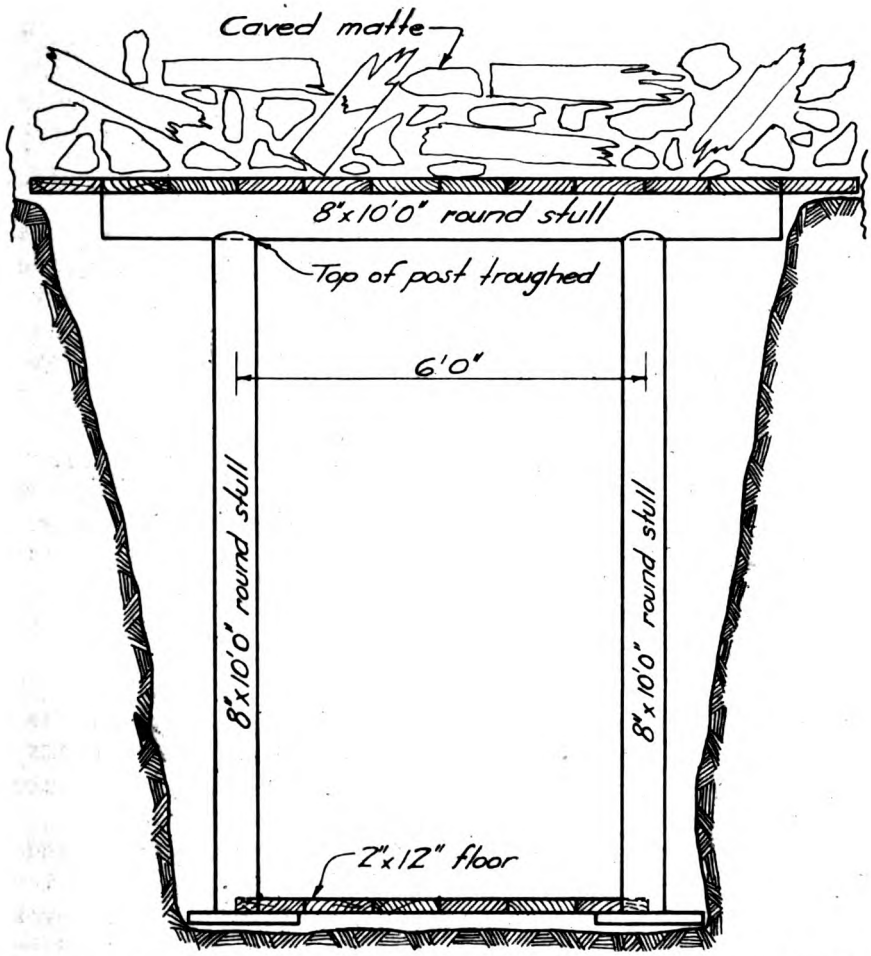

Figure 13.- Typical top-slice timbering, used as shown in advance, at intervals of 5 feet. Extended laterally in stopes

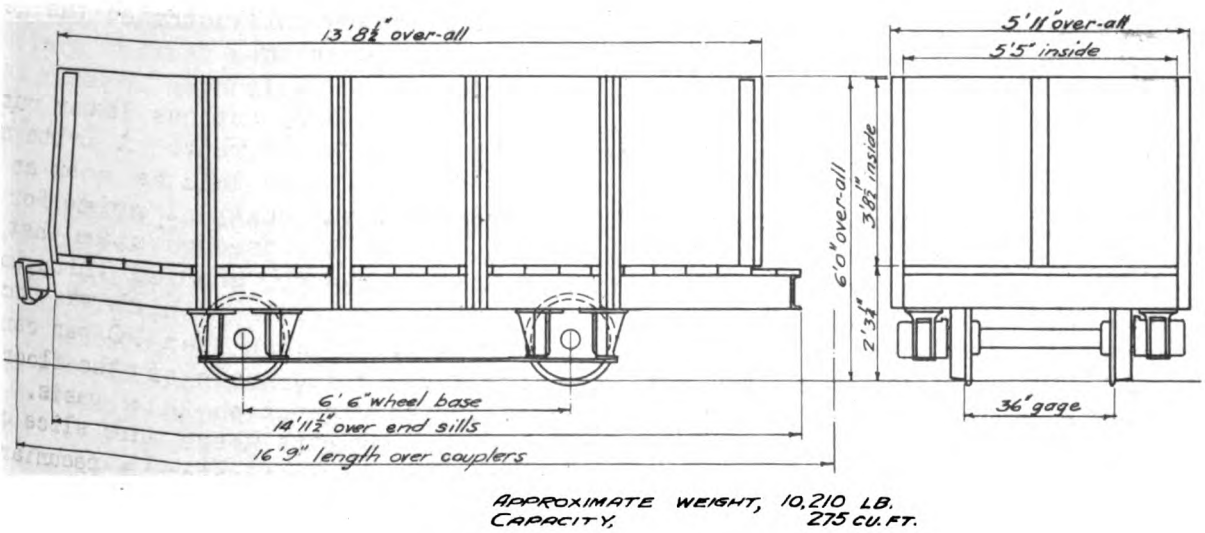

Figure 14.- Mine car

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San Pedro Orebody

Open Stopes.--Mining in the San Pedro orebody was confined to small operations that could be carried on without weakening the ground to the extent of jeopardizing the Catillas glory-hole operation. The stopes may be described as small underground glory holes, so planned as to reduce shoveling and tramming to the minimum. Adequate pillars were left to support the ground. The western and northwestern extremities of the orebody were caved and drawn to completion after the limit of extraction by open-stope methods had been reached.

More than 70 per cent of the total ore in the orebody has been recovered, leaving the balance to be recovered by caving methods after completion of the Catillas glory-hole operation.

UNDERGROUND HAULAGE

0-meter Level

The locomotives on the 0-meter level are of the trolley type, weigh 8 tons, and use 250-volt current. Ten cars of the type shown in Figure 14 made up the usual train. The average length of haul for the Hilltop orebody was approximately 1,800 feet. This equipment is now used for hauling ore from the General shaft to the crushing plant, a distance of 500 feet.

Both cars and locomotives have given excellent service as a result of adequate strength of design and a high standard of maintenance. After 10 years of service, during which time 3 locomotives and 22 cars have handled 11,100,000 tons of ore, cars and locomotives are as good as new from the standpoint of the service they give.

For transporting waste to the dump, a 4-yard side-dump car of the Western wheeled scraper type was used with excellent results. The waste dump was managed according to the general practice of large steam-shovel operations.

44 and 105 meter Levels

Tracks on the 44 and 105 meter levels are of 25-pound rail laid to 20-inch gage on 5 by 6 inch by 3 foot 6 inch pine ties spaced at intervals of 2 feet. Grades are 0.5 per cent in favor of the load.

Locomotives are similar to those used on the 0-meter level, and likewise use 250-volt current, but weigh only 4.5 tons. Cars are of the gable-bottomed type with a capacity of 70 cubic feet and a maximum door opening of 1 foot 6 inches. From 8 to 14 cars are handled in a train, depending on the length of the haul and the switching facilities at the different chutes. The average length of haul for the Catillas orebody was approximately 650 feet.

When full production was first reached in the Catillas orebody, two trains were operated on the 44-meter level and one on the 105-meter level. At a

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later stage one of the trains from the 44-meter level was moved to the 105-meter level and the remaining one was used principally for handling waste from the north side of the Catillas orebody. In the final stage, after the stripping of waste was nearly completed, all of the equipment was concentrated on the 105-meter level but only two trains were operated. The haulage was much more efficient when done on two levels, because of less interference of the trains. When operating at the 105-meter level, on a three shift per 24 hour basis, in dry weather and when loading conditions are otherwise fair, two trains of from 10 to 12 cars each have a demonstrated average capacity of more than 3,500 tons per day.

165-meter Level

Track specifications were the same on the 165 as on the 105 meter level. Power was supplied by a 4-ton, storage-battery locomotive. The batter was of the Edison alkaline type, had 84 cells, and a deliverable current capacity of about 300 ampere hours. A charging station was maintained on the level. Tramming was on a 2-shift basis, and two batteries were maintained for the locomotive. Cars were of the rocker-dump design, with a capacity of 35 cubic feet. From 8 to 12 cars were hauled in a train.

HOISTING

Ore is hoisted in the General shaft. The skips are of 135-cubic foot capacity, equivalent to 5 metric tons. They have cast-steel crossheads with single safety dogs of special design, and weigh about 4 tons. Sheave wheels are of the bicycle type, 8 feet in diameter. Cables are 1-1/8-inch diameter, and are right-hand Langlay, special scale construction, of the best obtainable quality. The hoisting speed is 800 feet per minute.

The hoist is of a double-drum, post-brake pattern, driven by a 450-hp., 2,200-volt motor through a single reduction of herringbone gears. The brakes are controlled by oil under pressure, and are counterweighted to set automatically when the oil pressure is released. They are connected through a solenoid trip to Lilly controls, which automatically cut off the electric current and allow the brakes to set, in case of overspeeding or overwinding. The drums are 6 feet in diameter and are not grooved.

The skips discharge into a small steel transfer chute in the headframe, which is equipped with a heavy baffle to break the fall. The ore is loaded from the transfer chute into a shuttle car which travels on a trestle parallel to the long axis of the shaft and dumps into any one of the five storage bins. This car is of the gable-bottomed type, and has a capacity of 150 cubic feet. It is operated by a small electric hoist mounted at the and of the trestle. The five storage bins have chutes along the 0-meter haulage-way (fig. 8), equipped with check gates and underswung cut-off gates exactly like those used in the glory-hole loading stations on this level.

These bins provide a surface storage capacity of more than 60 per cent of the daily production. Tramming on the 0-meter level is thus nearly, if

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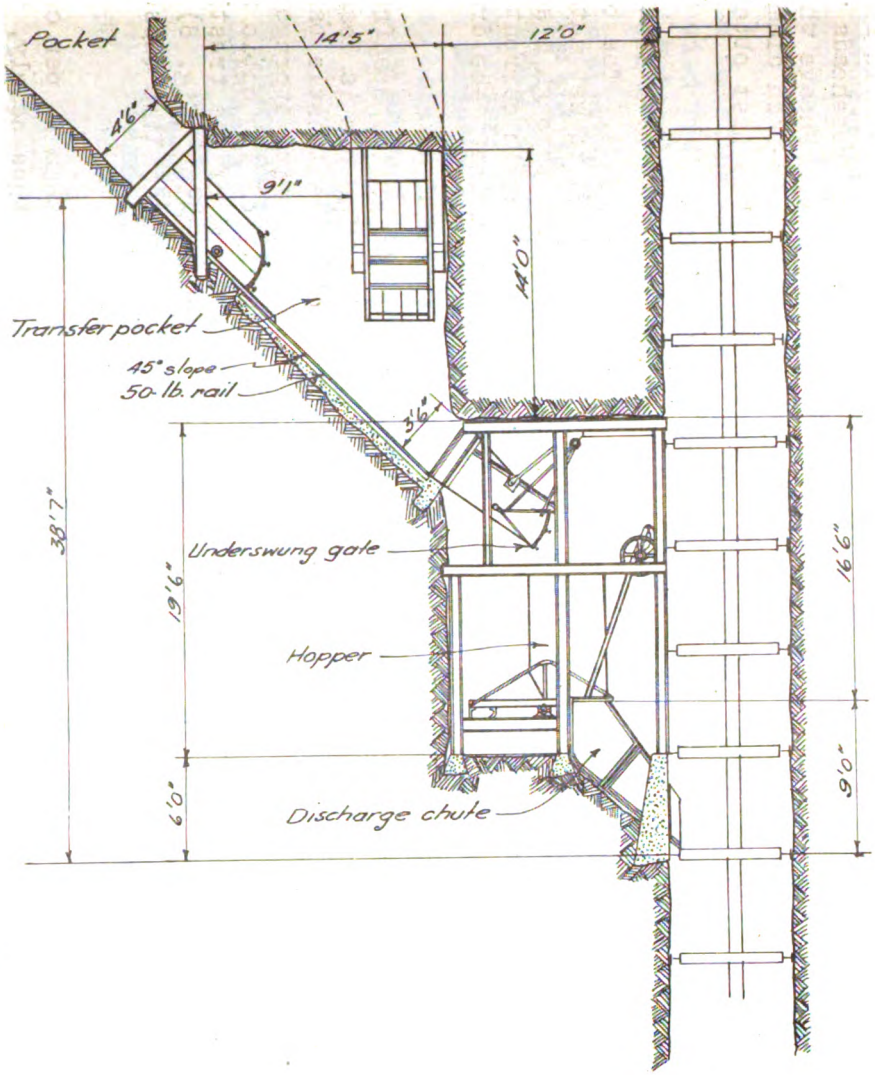

Figure 15.- Projection of loading pocket, 105-meter level

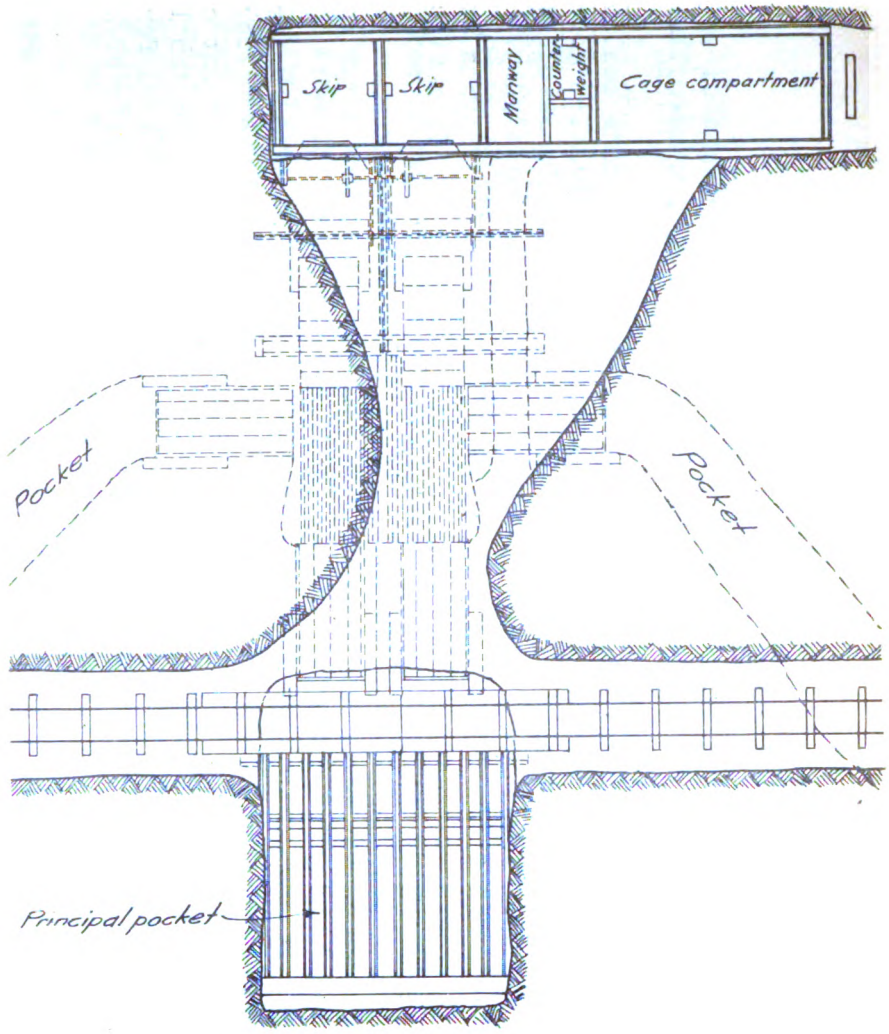

Figure 16.- Plan of loading pocket, 105-meter leve

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not quite, independent of the hoisting, and can be regulated to achieve the maximum efficiency of the trains and the crushing plant at the mill.

Ore from each orebody can be kept separate until delivered to the crusher. As the only reliable production sample that can be secured is the one cut by an automatic sampler at the mill after crushing, the separation of the ore until after this sample has been cut has been very valuable in furnishing a reliable check on the grade of ore from each of the several orebodies.

Figures 15 and 16 show the arrangement of pockets and skip-loading mechanism serving the 105-meter level. This installation was made in 1924, and still gives excellent service. The upper transfer pocket is essential where ore from different sources is to be kept separate. The hoppers are easily manipulated and rapid in action. The average halted time of the skip while receiving its load is 6 seconds. The chief criticism of the installation is that it operates with considerable violence, with a resulting high cost of maintenance. The retardation of the action by means of springs or counterweights would be desirable, but no practical system has been devised to secure this result. Access to the control gates of the upper pockets is had by way of a small crosscut from the shaft manway.

Figure 17 shows the corresponding installation to serve the 165-meter level. The hoppers were given a smaller horizontal cross section and a slope of 60° to render them less violent in action. The mechanism retains all the other features of the other installation. It is slightly slower, operates more smoothly, requires less maintenance, and is considered a better installation.

The hoisting capacity of the shaft has never been demonstrated over a long period but is estimated to be more than 1,700 tons per 8-hour shift from the 105-meter level pockets.

The service cage is 12 feet long and 5 feet wide. It weighs approximately 4 tons and is counterbalanced by a 5-1/2-ton counterweight. The hoisting speed is 600 feet per minute. The cage hoist is driven by a 150-hp., 2,200-volt motor through a single reduction of herringbone gears. The drums are 6 feet in diameter and are grooved. Sheaves and cables are the same as used on the skip hoist. Brakes are hand operated. The hoist is equipped with a Lilly control to prevent overspeeding and overwinding.

WAGE, CONTRACT, AND BONUS SYSTEMS EMPLOYED

The contract system has been generally in force on all parts of the work where measurable units of output for small groups of men can be accurately measured. Practically all development, preparation, and ore breaking are on this basis. Timbering of routine or standard nature is contracted wherever possible. Tramming, hoisting, and servicing of various kinds are done on day's pay.

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Contract prices are carefully fixed, usually by the foreman directly in charge of the work, and are changed as seldom as possible. Good work is recognized, and is not penalized by price reductions below the average basis. Certain contractors, recognized as capable managers, always have high earnings, while others are continually "scraping bottom." The latter are given the benefit of special supervision, and every effort is made to develop them into good managers. Hopeless cases are eliminated as soon as they are recognized. The contractors mostly are Mexicans, with a sprinkling of Italians and Greeks.

Contractors and men are guaranteed a minimum daily wage, which they may draw daily from the time office if they wish, such payments being charged against the contract. Contracts are liquidated four times each month. The balance due, after all deductions, is paid directly to the contractor, who personally distributes excess earnings among the men in such proportion as he sees fit.

The independent attitude of the Fresnillo workmen holds the contractors well in check; those who do not share a reasonable portion of their excess earnings with their men soon become known, and the better men will not work for them.

The company maintains standards of performance inflexibly; drifts and raises must be full size, waste must not be broken in the stopes, etc.; but to get these results, supervision must be close and discipline firm. Work not executed in an acceptable manner will not be paid for in full until the defects are made good.

The contractor is always a man of higher than average intelligence, and amenable in some degree to the company's viewpoint. With his incentive of personal gain, and with proper handling, he supplies a medium through which close supervision of the work can be secured.

For about three years, 1924-1927, an extensive trial was given to the bonus system, in the belief that it would lead to a more general and even distribution of earnings among the men, one more nearly proportioned to their individual efforts, and in this way promote individual efficiency. The men were organized in the smallest possible groups, each group under a petty boss. Standards of output were established from records of past work done on a contract basis. The system was carefully explained to the workmen, and was under the administration of men of ample experience. Although good results were obtained at the start, the system finally failed, partly because of peculiar characteristics of the workmen. A week when bonuses were good would invariably be followed by one of slackened production and much laying-off on the part of the men. When bonuses were low, discontent prevailed. The bonus came to be looked on merely as extra pay, and the local unions began to make it a subject of discussion. When results were poor, there was no direct and forceful way of handling the situation. It is possible, however, that the failure of the bonus system was due to the following three principal causes: (1) The widely spread labor unrest that was sweeping the

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country at the time; (2) insufficient supervision, due to the diversity of the operations and the wide areas covered by the foremen; and (3) the failure to enlist the entire cooperation of petty bosses and group leaders. Many of these latter became successful contractors when the contract system was restored.

VENTILATION

Natural ventilation of the workings has always been adequate, because of the existence of many old workings to which connections were made at frequent intervals.

FIRE HAZARDS

The principal fire hazard in the mine has been the top-slice area. Practically all of the workings in this area were timbered and the mat itself, of course, was an ever-present danger. It was recognized that a fire here would be disastrous. All of the workings in and around the area not actually required for ventilation or extraction of ore were tightly sealed off with adequate masonry bulkheads. Fire doors were constructed at stragetic points so that gases could be controlled in the event of fire. Fire extinguishers were kept at handy places in the area, and bosses and contractors were instructed in their use. Connections were made from the main air lines to the pump columns in the nearby pump shaft so that a large volume of water at high pressure would be at once available if required. The area was patroled by inspectors at all times when men were not working there; that is, on the graveyard shift and on Sundays and other holidays.

A large number of men were trained in the use of oxygen breathing apparatus, and four crews of five men each were kept in continued training. Demonstrated ability to use the apparatus has been a requisite for employment of a foreign boss in the mine. A smoke room was provided to make the training as effective as possible. Fire drills were held in the top-slice area at least once each month.

The top-slice operation was carried through to conclusion without a serious fire. The only threat of fire in the whole operation was a very small and smouldering blaze that resulted from blasting timber in a well-ventilated part of the slice. A wide-awake Italian boss extinguished the fire in a few minutes with the contents of three fire extinguishers. The area for 10 feet or so around the blaze was at once re-opened by spiling to insure that no sparks remained, and the area was kept under close observation for several weeks afterward, but nothing further occurred. The incident served as a warning of the ever-present danger, and gave a practical demonstration of the high value of careful and well-understood directions as to what to do in case of fire.

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FIRST-AID ORGANIZATION AND ACCIDENT PREVENTION

Because of the extreme likelihood of the infection of open wounds in this country, first aid to injured men in the mine is reduced to the minimum of simple bandages, splints, and tourniquets, and a large number of workmen and all of the bosses have been trained to give this treatment. The policy is to get the injured man to the hospital with all possible dispatch. In serious cases, when it would be dangerous to move the injured man, the doctor or his assistant comes to the mine. Stretchers are kept at convenient places underground and first-aid material is kept in the principal magazines. A completely equipped first-aid station is maintained near the collar of the General shaft. The company maintains a complete and modern hospital.

Safety work is under the direction of a Mexican engineer. He has under him a corps of inspectors who are constantly in the mine, on the lookout for dangerous conditions. When found, such conditions are immediately reported to the boss directly in charge of the work. In urgent situations, the inspectors have authority to stop the work until the place has been inspected by the boss, who then assumes all further responsibility. The inspectors make daily reports to the safety engineer, who personally verifies special conditions and takes them up with the mine officials.

The use of goggles, hard-boiled hats, and shoes or sandals of special design is obligatory for all men underground. The use of canvas leggings is also required of men working on grizzlies or other places where leg protection is necessary. This material is supplied by the company at a price considerably below cost.

Each lost-time accident is personally investigated by the safety engineer and the foreman in charge of the work, usually accompanied by someone of higher authority. If possible, the accident is reenacted to determine the personal responsibility. Reports are made to the management, and are followed up in detail. Lay-offs are now applied to experienced men who are injured by their own carelessness, and to contractors, bosses, and others who countenance violation of safety regulations. This measure was adoptai as being the only way of influencing those men who are not amenable to other methods; a thorough investigation is made of each case before laying off a man.

All bosses are paid a monthly safety bonus based upon the total shifts worked under their supervision and the number of accidents charged to their organizations.

Once each month the mine workings are thoroughly inspected by committees of workmen, accompanied by the safety engineer or his assistant. A written report is made of all conditions thought to be unsafe, and these reports are discussed at a monthly meeting of the committees, all of the foreman and bosses, and the mine officials. Each recommendation is given thorough consideration, and a definite decision is promptly reached. It is then the duty of the safety engineer to follow up on each accepted recommendation and see that is is carried out.

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The contractors are called together once each month and the accident record for the preceding month is discussed with them in detail, particularly with reference to the accidents that occurred among their men and to measures that can be taken to avoid similar accidents in the future.

A general safety committee, comprising the management, department heads, and principal assistants, meets once each month for a review of the record of the previous month and for the discussion of safety matters of a more general nature.

By far the greater part of the accidents that occur are comparatively slight in their nature and can be traced directly to the following causes: (1) Inadequate clothing; (2) personal carelessness; (3) lack of intelligence or plain torpidity on the part of the injured man. Injured men receive full time while incapacitated, so that many of them do not give sufficiently serious regard to a minor accident involving a few days lost time.

At present, a special effort is being made to bring home to every man his responsibility for his own safety. This is being done by (1) determining the personal responsibility for the accident in every possible case; (2) by personal lectures to the men who are injured, by the safety inspector, the foreman involved, or in more serious cases, by the mine officials; and (3) by discipline in the form of lay-offs of from three to five days for injured men who are convicted of carelessness or indifference.

It is probable that future progress in accident prevention at Fresnillo must be along these lines.

Accident Statistics

Complete statistics to cover the early years of the operation are not available. For the year ended June 30, 1925, the accidents per 1,000 shifts worked in the Hilltop orebody were as follows:

Fatal	0.00
Grave	.03
Minor	.59
Total	0.62

The above figures cover all shifts charged to the Hilltop operation including transportation, surface, and general mine expense. Grave accidents were due to two principal causes: (1) Failure to keep the slopes of the glory holes carefully barred down; and (2) failure of workmen always to be tied with a rope when working in steep and dangerous places.

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For the six years from July 1, 1925, to June 30, 1931, ore-breaking accidents per 1,000 shifts worked for the two principal mining methods were as follows:

	Glory holing	Top slicing
Fatal	0.0044	0.0048
Grave	.0286	.0095
Minor	.3722	.6762
Totals	0.4052	0.6905

The above figures apply only to ore breaking and do not include transportation, surface, or general mine expense. They exclude also accidents that do not occasion a loss of time for the injured man.

For divisions of the work other than ore breaking, the accident statistics for the oxide divisions can not accurately be separated from the total for the mine. The following tabulation of total accidents per 1,000 shifts worked for the whole mine, including the sulphide divisions, is therefore presented as showing the general trend.

Year ending June 30	Fatal	Grave	Minor	Total
1926	0.02	0.01	1.50	1.54
1927	0.01	0.01	1.42	1.44
1928	0.03	0.02	1.31	1.36
1929	0.03	0.06	1.26	1.35
1930	0.01	0.04	0.86	0.91
1931	0.03	0.03	0.62	0.68

The decided improvement in the records for the last two years is the result of a determined effort to reduce the number of accidents and to interest the workmen in their personal safety. The effects of a safety campaign are cumulative to a considerable degree. It has been difficult to establish a trend toward improvement, but once established, it is expected to continue.

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Table 1. - Hilltop orebody

Glory-hole method. 3,858,688 tons of ore.

July 1, 1921 to June 30, 1925. 67,092 tons of waste stripping.

A - MINING COSTS (in U. S. dollars per short ton of ore)

	Labor	Super-vision	Power	Explo-sives	Timber	Other supplies	Total
Development ...	$0.0179	$0.0024	$0.0018	$0.0040	$0.0023	$0.0069	$0.0353
Ore breaking ..	.0574	.0032	.0190	.0510	.0009	.0182	.1497
Secondary blast	.0193	.0008	.0016	.0021	.....	.0014	.0252
Transportation.	.0252	.0016	.0058	.....	.....	.0106	.0432
General under-ground expense	.0050	.....	.0007	.....	.....	.0018	.0075
Surface expense	.0035	.....	.....	.....	.....	.0006	.0041
Totals	0.1283	0.0080	0.0289	0.0571	0.0032	0.0395	0.2650

Per cent of total cost

Labor and supervision 51.5
Power and supplies 49.5

B - MINING COSTS (in units of labor, power, and supplies, per short ton of ore or per foot of advance)

Ore production

Labor (man-hours per ton)

Ore breaking	0.332
Secondary blasting	.129
Transportation	.129
General underground exp..	.023
Surface expense	.014
Total	0.627
Tons per 8-hour man-shift	12.76

Power (Kw.h. per ton)

Ore breaking	1.41
Secondary blasting	.12
Transportation	.43
General underground exp..	.05
Total	2.01

Supplies

Explosives, lb. per ton..	0.217
Timber, bd.ft. per ton ..	0.070

Development (313 tons mined per foot of advance)

Labor, man-hrs. per ft...	32.14
Power, Kw.h. per ft.	41.5
Supplies:	
Explosives, lb. per ft.	5.04
Timber, bd.ft. per ft..	15.9

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The waste-to-ore ratio in the Hilltop orebody was 0.017 to 1. As the foregoing costs are calculated on the basis of tons of ore, the figures should be divided by 1.017 to obtain the cost per ton of material mined.

Table 2. - Orebodies other than the Hilltop

Orebody	Mining method	Period	Tonnages
Catillas and San Nicolas	Glory hole		4,544,914[1]
Pilar	Top slice		436,500
San Pedro	Miscellaneous	16 mos. 1926 and 1927	130,000

A - MINING COSTS (in U. S. dollars per short ton of ore)

	Labor	Supervision	Power	Explosives	Timber	Other supplies	Total
Development	$0.0125	$0.0107	$0.0075	$0.0126	$0.0171	$0.0383	$0.1887
Ore breaking:							
Glory hole ...	.1303	.0179	.0290	.0584	.0037	.0238	.2631
Top slice	.6179	.0179	.0707	.0574	.4100	.0258	1.1997
Miscellaneous.	.3673	.0179	.0707	.0654	.0045	.0242	.5500
Transportation..	.0874	.0071	.0181	.....	.....	.0269	.1395
Gen. underground expense	.0659	.....	.0140	.....	.....	.0763	.1562
Surface expense.	.0089	.....	.....	.....	.....	.0015	.0104

1/ Plus 1,328,360 tons of waste stripping.

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Table 2. - Orebodies other than the Hilltop--Continued

B - MINING COSTS (in units of labor, power, and supplies, per short ton of ore or per foot of advance)

Ore production

Labor (man-hours per ton)

Ore breaking:	
Glory hole	0.800
Top slice	3.830
Miscellaneous	1.664
Transportation	.398
General underground expense	.311
Surface expense	.009

Power (kw. h. per ton)

Ore breaking:	
Glory hole	2.13
Top slice	5.22
Miscellaneous	5.22
Transportation	1.33
General underground expense	1.03

Supplies

Explosives (lb. per ton):	
Glory hole	0.343
Top slice	.338
Miscellaneous	.385
Timber (bd. ft. per ton):	
Glory hole	.082
Top slice	9.151
Miscellaneous	.010

The waste-to-ore ratio in the Catillas and San Nicolas orebodies was 0.292 to 1. As the ore-breaking costs given above for these orebodies (under "Glory hole") are calculated on the basis of tons of ore mined, those figures may be divided by 1.292 if it is desired to obtain the costs per ton of material mined. The costs shown for "Transportation," being calculated on an ore basis, may likewise be reduced by dividing by the factor 1.13 to obtain costs per ton of total material handled.

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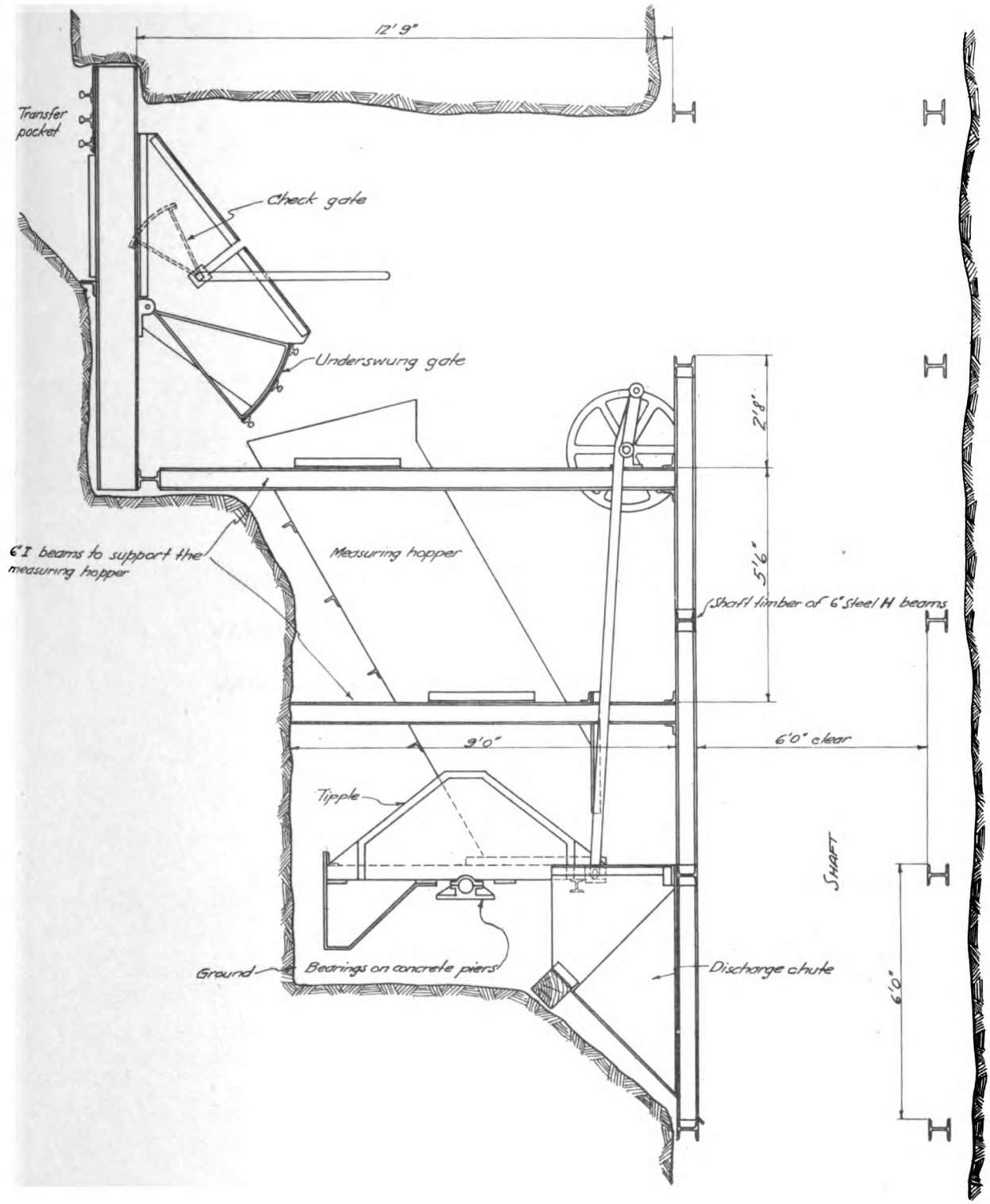

Figure 17.- Projection of loading pocket, 165-meter level

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